Metallurgical and Chemical Engineering

An Incorporation of
Electrochemical and Metallurgical Industry—Iron and Steel Magazine

Vol. XI. No. 3 New York, March, 1913. 25 cts. per Copy. $2.00 per Year to U. S. and Mexico. $2.50 per Year to other Countries.

Index to Advertisers, page 71. Classified Directory, pages 67, 58, 69

METALLURGICAL & CHEMICAL ENGINEERING

ISSUE OF MARCH, 1913

METALLURGICAL & CHEMICAL ENGINEERING
239 WEST 39th STREET, NEW YORK

Metallurgical and Chemical Engineering

A Consolidation of ELECTROCHEMICAL AND METALLURGICAL INDUSTRY and IRON AND STEEL MAGAZINE.

VOL. XI — NEW YORK, MARCH, 1913. — NO. 3

PUBLISHED MONTHLY BY THE

McGraw Publishing Company, Inc.

JAMES H. MCGRAW, President.
C. E. WHITTLESEY, Secretary and Treasurer.
239 West 39th Street, New York.
TELEPHONE CALLS, 4700 BRYANT. CABLE ADDRESS, METCHEM, NEW YORK.

WESTERN OFFICE....................421 Boston Bldg., Denver, Colo.
CHICAGO OFFICE....................Old Colony Building
PHILADELPHIA OFFICE....................Real Estate Trust Building
LONDON OFFICE....................Hastings House, Norfolk St., Strand

E. F. ROEBER, PH.D., Editor.
H. C. PARMELEE, Western Editor.

J. M. MUIR, Manager.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; *for all other countries,* $2.50. (*European exchange,* 10 *shillings,* 10 *marks,* 12.50 *francs.*)

Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879.

NEW YORK, MARCH. 1913.

CONTENTS.

More Aspects of Conservation

We have often puzzled over the meaning of the stock apology of those publicists who advocate the restriction of water-power development on the ground that such powers "should be reserved for the people." Naturally enough, they do not as a rule trouble to explain exactly what the phrase means, if anything. Our thanks are therefore due to Secretary Stimson, who, at a hearing before the Foreign Affairs Committee in regard to the restriction of water-power development at Niagara Falls, made a dangerous descent to the definite. Advocating that the water should be developed to its fullest efficiency, he suggested at the same time that provision be made to give preference in the power permits to cities and to lighting and other public service corporations.

We could wish that it had been somebody with a lesser reputation for clear-headedness than Secretary Stimson who had made this suggestion. From an economic point of view, it verges on unsoundness, since it is doubtful whether in unrestricted competition for hydro-electric power the consumptive outlets favored by Mr. Stimson could, in the long run, hold out against outlets which, on account of their much superior load factor, form the natural market for such power. As is well known, the annual load factor (that is, the ratio of the average load to peak load during the year) of municipal lighting, and other similar public-service load, is usually in the neighborhood of 30 per cent. The much higher investment per h.p. capacity in an hydro-electric plant, as compared, for instance with a steam generating plant, results in a diminution in the difference of cost between hydro-electric power and steam-generated power as the load factor diminishes, and it is safe to say that in most localities the difference in cost on service of low load factor is not such as would be appreciable to the average consumer of such power.

What we wish particularly to call attention to, however, are the really remarkable aspects of Secretary Stimson's suggestion from the straight and sincere point of view of conservation. We are not going to touch, on this occasion, on the monumental waste of power at Niagara Falls enforced (for the present) by the treaty between the United States and Great Britain, but confine ourselves to the consideration of the 20,000 cubic feet per second permitted for American use by the treaty. This represents a definite amount of power. Even Secretary Stimson appears to appreciate this, and is sufficient of a conservationist to desire that it be developed "to its fullest efficiency," whatever that may be. It never seems, however, to have occurred to him that the principles of conservation demand, in the first place, that the power by *utilized* to its fullest efficiency.

Seventy per cent of all hydro-electric power which is devoted to municipal purposes is wasted; wasted, because not

utilized. And it is wasted irretrievably; it is not a resource which, like the natural fuels, if not utilized today is saved for tomorrow. It *might* have been utilized, and by its utilization have saved a corresponding amount of exhaustible fuel. This makes the crime a double one, one of omission and one of commission. The only possible excuse for the consumption of hydro-electric power in municipal and similar service is the lack of a market with better load factor. Such a market is furnished in an ideal form by the electrochemical industries, which can be so run as to have a load factor of from 95 to 100 per cent.

In view of the power famine in the Niagara district, Secretary Stimson will hardly offer this excuse, however, for his astonishing proposal. Ignorance of the elementary engineering proposition which we have pointed out above is almost inconceivable in one who has at his disposal the advice of the Engineering Corps, unless, perhaps, the natural activities of the Department of War, dealing as they do with destruction and, in a minor degree, with construction, are unfavorable to considerations of mere conservation. We are reluctantly forced to the conclusion that, in making the suggestion, Secretary Stimson was influenced, probably against his better judgment, by the unreasoning clamor of ill-informed persons whose prejudiced vociferations have done so much in recent years to retard, against every principle of conservation, the development of this great branch of the country's resources.

The conservation of our exhaustible supplies of natural fuels demands the development of the water powers of America, and every attention should be paid to their efficient utilization, not merely in relation to efficiency of power production, but particularly in relation to the even more important matter of efficient consumption in respect of load factor. It is becoming increasingly evident that the political and professional advocates of conservation cannot be trusted, either because of their ignorance or because of their prejudice, with questions of water-power policy; and the engineering professions and technical press will fail in their duty if they do not supply this deficiency by furnishing persistent instruction and criticism.

Pumping Slime with the Air-Lift

The growing popularity of the air-lift for transferring slime in connection with the cyanide process, calls to mind the fact that data and information pertaining to that operation are comparatively meagre. As a water pump the air-lift has been the subject of several investigations, with the result that definite conclusions have been announced as to its advantages and disadvantages, efficiency and essential features of design. The adaptation of the air-lift to slime pumping has been based largely on the data derived from water pumping, and in this respect it is probable that no serious error has been committed. Nevertheless, the idea suggests itself that a mechanical investigation of the air-lift as a slime pump would be valuable to the metallurgical engineer.

It is true that such data of operation as air and power consumption can be obtained from individual sources, but usually they refer to the performance of particular pumps which may or may not be well designed or operating under favorable conditions. In any event they are not comprehensive enough to form a basis for conclusions. An independent experimental investigation ought to develop such data as are not likely to be brought out in practice, and determine relations having a direct bearing on correct design and operation. There are five variables which may affect a particular size and type of air-lift: submergence, lift, discharge, volume and pressure of air. To test various combinations of these variables involves a vast amount of work, but some interesting and valuable data would result.

It is unlikely that the efficient percentage of submergence in pumping slime would vary greatly from that determined for pumping water, namely, 63 per cent; but there are other relations in which the denser medium would make a difference. The coefficient of pipe friction and slip; the quantity of air for maximum discharge; the ratio of volume of air to volume of slime; maximum output; efficiency and input, and other points which could be determined experimentally under varying conditions of submergence and lift, would form a basis for intelligent design and operation of air-lifts for slime. No harm would result if several such investigations should be undertaken at the same time by different parties; on the contrary, the possibility of having several sets of data for comparison would be rather attractive.

Requirements for Successful Ore-Flotation

Signs of the times in metallurgy point toward a wider adoption of flotation in copper and zinc-lead concentration in the United States. This form of concentration, developed to its highest state of perfection in Australia, and widely used there in the treatment of zinc-lead ores, has been the subject of widespread experimentation during the past year. Two notable installations have been made in the western hemisphere; one at the Braden copper mines in South America, and the other at the Butte & Superior zinc mines in Montana. Experimental plants have been built and used at several places and many ores have been tested. The success which has attended these first efforts evidently has influenced the management of the Miami and Inspiration copper companies in Arizona to delay completion of plans for remodeling the first-named mill and constructing the second, until tests can be made to determine the possible value of flotation in the treatment of their ores. The Federal company in Idaho also is reported to be experimenting with the process on zinc-lead ores.

The requirements for successful ore-flotation have been pretty well determined, and those who contemplate experimenting with the process should inform themselves on these points before proceeding further. Decomposed, oxidized ore, in general, seems to exert a deleterious effect, forming no "froth" and yielding poor results. Clean sulphide minerals are most favorable and give good results. Close sizing seems to be a prerequisite, for large particles of mineral in an unsized feed show a tendency to fall through the scum of floating concentrate, and in so doing they not only pass into the tailing, but carry finer mineral with them. This fact suggests a maximum size of particle, which should be determined by experiment.

The process further requires uniform conditions in the feed, and in the speed of the flotation machine. Irregular

quantity of feed disturbs the operation and causes loss of mineral. Fluctuation in the speed likewise has a bad effect and should be overcome by providing an independent source of power. The supply of oil should be constant. Not the least important factor is the consistency of the feed. Australian practice has developed a ratio of water to solids of 3 or 4 to 1. In some cases a thicker pulp has been used, but more dilute pulps are not advantageous.

The excellent paper by Mr. H. Hardy Smith, presented before the Australasian Institute of Mining Engineers and published in this issue, may be recommended to the careful attention of all interested engineers as one of the most valuable contributions to the literature on flotation.

Nichols-Hesse Dinner

Those who attended the International Congress of Applied Chemistry last year in New York City will certainly be delighted to hear of a movement to show the appreciation of their fellow chemists to the two men who bore the brunt of the battle—the President of the Congress, Dr. William H. Nichols, and the Secretary, Dr. B. C. Hesse. This mark of the appreciation of their work will take the form of a complimentary dinner, to be held at the Chemists' Club on April, 1912. It is particularly pleasant that this decision should have been reached by the joint unanimous action of the executive officials of the various chemical societies. Without doubt the rank and file of American chemists will join to make the dinner a brilliant success.

Can the Bessemer Steel Industry Be Saved?

It is an interesting fact that there has been no important improvement in the quality of Bessemer steel since the early years of the industry, now more than half a century old. Improvements in processes have been made, but they have conduced to lower costs rather than improved quality. Nothing could be much more beautiful, from an economic standpoint, than the method by which molten pig iron at a red heat is converted in a few minutes into molten steel at a white heat, yet all the trends of the times were apparently in recent years dooming this industry to eventual extinction. The United States saw its maximum production of Bessemer steel as far back as in 1906, whereas the iron industry as a whole has so grown meanwhile that pig iron is being produced at a rate 30 per cent greater than in 1906. In one product after another Bessemer steel has become unacceptable. Producers assert that in many cases the preference expressed by buyers for open-hearth steel is unreasonable, but they do not take steps to correct what they denominate a misconception.

The failure of Bessemer steel to equal open-hearth steel in quality is a failure under unfortunate circumstances, for the raw material of the Bessemer process is by far the purer. Two facts plead that someone should save the Bessemer process, one being that the United States possesses reserves of hundreds of millions of tons of relatively pure iron ore, within the Bessemer limit as to phosphorus, while the other is that in many cases, particularly the rolling of sheets, the drawing of wire and the threading of pipe, Bessemer steel yields itself more readily to the mechanical operation than does open-hearth steel.

Two losses confront the industry. One is, perhaps, only a paper loss, covering the greater value which a few years ago Bessemer ores lying in the ground were held to possess over non-Bessemer ores, apart from the difference in average iron content, while the other lies in actual equipment which is being rendered less useful. The loss is not merely in Bessemer vessel capacity, and indeed that has been saved in some instances by the adoption of the duplex process, but covers also the complete rearrangement of works, if open-hearth is to be substituted for Bessemer, by reason of the much greater ground space required by open-hearth furnaces of equal tonnage output.

There is no doubt that Bessemer steel possesses certain valuable qualities as compared with open-hearth, even though they cannot all be measured quantitatively, and Bessemer steel, merely shorn of certain objectionable qualities, would undoubtedly prove superior to open-hearth for many uses involving an enormous tonnage.

We have expressed the opinion before that the savior of the Bessemer converter industry will be the electric refining furnace, and nothing has happened that would induce us to change this opinion. The refining of molten Bessemer metal in the electric furnace will undoubtedly solve all technical problems, in as much as it will yield a metal that should satisfy all requirements. The reason why the industrial development of the addition of electric refining furnaces to existing Bessemer plants has been so very slow is a commercial one and a perfectly natural one. It concerns both the steel consumer and producer and has to do in both cases with the question of cost.

In the case of the consumer, the railroad companies naturally hesitate to pay a higher price for rails made from electrically refined steel, until the evidence of the superiority of the "electric steel rail" is complete. Now this complete evidence can only come from the experience with electric steel rails in the track for a series of years. That is a safe and sane, but naturally very slow development. Last May Mr. William R. Walker stated officially for the U. S. Steel Corporation before the American Iron and Steel Institute that at that time approximately 5600 tons of standard electric steel rails were in service in the United States and had then been in the track about two years, being exposed at times to severest weather conditions and to all the possible conditions of severe service, and that up to that time they had not heard of any basic electric rails in use in this country being broken in service. It is a pleasure now to say here that what was true then holds good now.

As to the producer, the attempts are being continued to reduce the additional cost of electric refining as much as possible. And this again is essentially a slow process, as it involves a lot of experimental work both as to the best division of the metallurgical reactions between converter and electric furnace and as to administrative measures and arrangement of plant. The goal is to get the cost of combined converter and electric furnace process down to the cost of the open-hearth. There is every reason to expect that this goal will be reached. When this is accomplished, it is evident that the Bessemer converter industry has been saved.

Readers' Views and Comments

The Tungsten-Molybdenum Couple

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have read with great interest Dr. Northrup's article on the tungsten-molybdenum thermocouple in your January issue, page 45, and also the discussions of Drs. Hering and Thwing on pages 66 and 67 of the February issue. It may interest these gentlemen to hear that the tungsten-molybdenum couple was announced three months before the meeting of the Philadelphia branch of the A. I. E. E. referred to by Dr. Hering.

My Congress paper on the "Applications of Ductile Tungsten" was reported in abstract in the September 12, 1912, number of this journal, page 580. It is interesting to note that our value for the maximum point is 540 deg., compared to 530 deg., reported by Dr. Northrup.

Colin G. Fink.

Harrison Laboratories, General Electric Company,
Harrison, N. J.

* * *

Conservation and the Wastage at Niagara

To the Editor of Metallurgical and Chemical Engineering:

Sir:—With reference to recent articles in your journal on conservation and the wastage at Niagara, I send you the following verses, which were submitted to the *Outlook* five years ago.

It was my idea that the *Outlook's* views on conservation and on industrial democracy would naturally make the lines acceptable.

They were promptly declined.

Niagara Captive

Niagara captive! And by ribbons led!
His mighty force with that of toiling head
And hand to join. So changed since ancient days
When red men chanted hymns of praise;
In flower-laden white canoe
Each spring their fairest maiden sent into
The Thunder of the Waters.

Niagara an adult and to Effort bred—
No more to play the livelong day,
But proudly share the sweat and grime
Of stalwart manhood's laboring prime.
The evergrowing purpose runs:—
Earth's wealth is measured, not the sun's;
The stewards of great treasure may
Not waste Tomorrow's dire need
For Pleasure's or for Profit's greed.

Oh, Hercules, still at thy labors keep!
Canst take the raging current from the flood
And swiftly, silent 'round a cable sweep?
Ye Seven Wonders of the ancient world,
Long since into oblivion hurled,
Your kings and gods born to commemorate—
'Tis to the people do we dedicate
The Wonders of Today.

Buffalo, N. Y. Edward Zaremba.

* * *

Early Mining Schools in America

To the Editor of Metallurgical and Chemical Engineering:

Sir: On p. 114 of your February issue occurs the following misstatement: "George A. Koenig * * * at the University of Pennsylvania gave the first course of mining that was ever given in any educational institution in the Uinted States."

Professor Koenig joined the faculty of the University of Pennsylvania in 1872, and by 1871 six educational institutions had already graduated classes in mining. The first classes in mining graduated at Columbia in 1867, and at the University of Michigan the same year, Massachusetts Institute of Technology in 1868, Washington and Lee in 1869, and Lehigh and Lafayette each in 1871. Among these early graduates were some of the most distinguished men in the mining profession.

A course in mining was also started at Rensselaer Polytechnic Institute in 1867, but it was abandoned in 1871 for lack of funds.

Who first taught a course in mining in the United States would probably be difficult to determine, for undoubtedly rudimentary courses in the art were given long before schools of mining engineering were organized, but certainly it long antedated Professor Koenig's day. Ex-Professor.

San Francisco, Calif.

* * *

Corrosion and Immersion Tests on Metals

To the Editor of Metallurgical and Chemical Engineering:

Sir:—It has been customary for years and there seems to be no inclination to depart from the custom of giving the results on corrosion and immersion tests in terms of percentage loss by weight.

This is obviously not good practice as two results are not at all comparable unless the ratio of the weight to the surface exposed is the same in both cases.

If a 1-inch solid cube of steel is immersed in a 10 per cent. solution of sulphuric acid for 10 hours and a 1-inch hollow cube of the same steel is immersed for the same length of time the actual loss in weight should be presumably the same and on calculating the percentage loss by weight as is customary, the results would be entirely different and erroneous.

The foregoing is given as an illustration and is perhaps exaggerated but illustrates the point.

The writer has adopted the plan of giving the loss in grams per square inch of surface exposed.

If it is desirable to adhere to the metric system grams per square centimeter can be given.

There is, of course, the factor of time and to compare two results the time must be equal. Unfortunately the results of several hours cannot be reduced to a unit, say the hour, as corrosion or dissolution does not proceed at a uniform rate. It would be well for investigations to adopt some standard as to time and I would suggest one day (24 hours), 7 days and 28 days as being reasonable.

Other factors of importance are:

(a) The metal should always be tested with the same acid or substance for which the metal is intended.

(b) The ratio of the acid to the test piece should be the same in comparative tests as where this is different one becomes neutralized more quickly than the other. A rule could be established whereby a definite number of cubic centimeters of the acid could be taken for each square inch of surface exposed on the test piece.

(c) A separate receptacle should be had for each test piece. Should two test pieces of different metals be placed in the same bath electrolytic action may set up and cause dissolution to proceed more rapidly than otherwise.

(d) The temperature of the bath should be equal on comparative tests.

(e) In reporting results it should be stated whether the test piece was cast and machined, cast without machining, rolled or forged, etc.

E. I. du Pont de Nemours Powder Company,
Henry Clay, Del.

Charles E. Arnold.

The Solubility of Alumina in a Bath of Fused Fluorides

To the Editor of Metallurgical and Chemical Engineering:

Sir:—As the following data were unknown to several men most prominently connected with the development of the aluminium industry when told them after the symposium on alumina held at the recent meeting of the New York section of the American Electrochemical Society and were found only recently by me in some casual reading, their reproduction may be of interest.

Deville[1], in a Memoir on Silicium, after giving the results of other methods of isolating that element says:

"Finally silicium may also be extracted from silica by means of the battery—an exceedingly simple process which may also be utilized for the electrolysis of all substances which dissolve under heat in the fluorides of the alkalies—and these substances are very numerous. A mixture is made of sodium and potassium fluorides in about equal proportions which are melted together above a double current burner fed by alcohol to which turpentine is added and aided by a blow-pipe. When the material is fully fused, calcined silica is fed in which quickly dissolves. Then, putting the platinum and carbon poles of a Bunsen battery of four cells in the crucible, silicium is seen to deposit on the negative pole and oxygen is evolved at the positive pole. I have most frequently been satisfied to verify the decomposition of silica by the battery by this experiment, in using, as negative electrode, a platinum wire which was transformed with great ease into a fusible silicide.

"I desired to generalize this method in applying it to very different materials which are soluble in the fluorides, first of all to isolate the elements by the battery, and then to determine their position in the electrochemical series. Thus one can deduce from the experiment which I have just described that silica separates into its elements before soda and potash, and in consequence that silica should be decomposed by the alkali metals, which is indeed the fact. But if for silica alumina is substituted, the decomposition by the battery, in the midst of the alkali fluorides, would be effected in quite different order. One obtains as a matter of fact, at the negative pole, sodium which burns with a yellow flame, and at the positive pole, fluorine, which is immideiately converted into hydrofluoric acid readily recognized by its odor. Alumina, therefore behaves quite differently from silica, and one must admit that it is not decomposed by the alkali metals, which all the attempts that have been made to obtain this reduction directly seem to have already shown."

In the light of subsequent events it is curious to note how close Deville, to whom we owe the first industrial isolation of aluminium, was, in 1857 to the winning of aluminium by means of electrolysis.

Had he been successful there was no cheap current available. It is also interesting to note that this was not an haphazard experiment but one undertaken with a view to generalizing. It is true that soda and potash should be decomposed more readily by the current than alumina, but the alumina should have been decomposed more readily than the fluorides, and no doubt would have been if the proportions had been propitious.

New York City. CHARLES A. DOREMUS.

* * *

Nitrogen, Its Influence in Steel

To the Editor of Metallurgical and Chemical Engineering:

Sir:—As several articles have recently appeared in the various technical journals relative to the momentous influence that small amounts of nitrogen may have in steel, and as various papers have been read on this subject before the engineering and chemical societies, the writer is led to relate the following from his experience and investigations dealing with the influence of this element in open-hearth and electric-furnace steels met with in every day practice, and of the grades used for small shafts, spindles, pneumatic tools, forming dies, etc.

In the main it may be conceded that nitrogen produces brittleness in all grades of steel though whether this element is itself the cause or whether its influence on other injurious elements, such as sulphur and phosphorus, in conjunction with it is the cause of brittleness, has not been satisfactorily determined.

It will require numerous tests, in view of the fact that practically similar conditions give inconsistent results, in which careful and complete analysis of a great many steels are compared with their physical properties to arrive at definite conclusions. It must be admitted, in general, that the results already obtained show practically no difference in the merits or demerits of steels that have been examined unless the phosphorus and nitrogen contents were abnormally high.

From the fact that metallurgists in common are at variance as to the proper methods for measuring the quality, or merits, of steel and from the ununiformity of the merit formulæ and figures, it is almost impossible to adopt a rule covering all conditions of comparison.

Mr. C. E. Stromeyer, in a paper read before the Institute of Naval Architects, London, England, gives the following rule for determining the quality, or merit, of steels from the phosphorus and nitrogen contents:

"The sum of the percentage of phosphorus plus five times the percentage of nitrogen should not exceed 0.080 per cent in good quality steel."

He further states, "that nitrogen has a ten fold greater effect than phosphorus in raising the tenacity of steel."

The formula most commonly used in England for determining the quality or merit of steels from their physical properties, elongation and tensile strength alone being considered as elastic limit and reduction of area are not factors, is as follows:

"M" = "T" + "E".
"M" = merit.
"T" = tensile strength in gross tons.
"E" = elongation in per cent.

If tensile strength and elongation are directly or indirectly a measure of toughness, or the combination of ductility and toughness and ductility or toughness are the opposite qualities of brittleness, then a comparison of results determined from formulæ combining tensile strength and elongation, and formulæ showing the effects of nitrogen, should give conclusive evidence relative to the influence of the element nitrogen.

The following table gives the complete analyses, including nitrogen, of a number of steels that were investigated by the writer to determine the influence of the nitrogen present:

TABLE No. I.—COMPLETE ANALYSIS OF SAMPLES

Test Number	Carbon	Silicon	Manganese	Sulphur	Phosphorus	Titanium	Chromium	Vanadium	Nitrogen	Duplicate Analysis on Nitrogen
1	.84	.17	.29	.009	.011	....	...	...	.012	.010
2	.68	.20	.30	.022	.013	.080	...	...	.008	.008
3	1.01	.10	.32	.009	.014	Trace	...	...	.013	.012
4	1.28	.15	.37	.009	.013	.010	...	...	.009	.009
5	.64	.12	.25	.022	.018	....	...	...	.003	.004
6	.48	.20	.19	.012	.008	....	.60	.20	.006	....
7	......	Not	deter	mined	......	......	......	......	.013	.014

Summary:

No. 1—Average of nine Electric-furnace heats.
No. 2—One Electric-furnace heat to which .50 per cent Ferro Titanium was added.
No. 3—One Electric-furnace heat to which .30 per cent Rutile (red oxide of Titanium) was added.
No. 4—One Electric-furnace heat to which .60 per cent Rutile was added.
No. 5—Average of nine Open Hearth furnace heats.
No. 6—One heat of Chrome-Vanadium steel.
No. 7—Drillings from Bessemer steel rail; only Nitrogen determined.

The method used for determining the nitrogen in these steels is as follows:

An Erlenmeyer flask of about 750 c.c. capacity is provided with a rubber stopper which carries a funnel and tube connected to a Bunsen condenser. The funnel is provided with a suitable stop cock.

About 500 c.c. of water, to which has been added 30 c.c. of

[1]Memoir on Silicium, Annales de Chimie et de Physique, 3, XLIX, p. 69, 1857.

a solution of caustic soda equal in strength to hydrochloric acid of specific gravity 1.12 are put in the flask and boiled. During this preliminary boiling one gramme of the sample steel is dissolved in 20 c.c. of hydrochloric acid and this solution is then put in the funnel of the Erlenmeyer flask from which it is allowed to fall, drop by drop, by means of the stop cock in the boiling alkali solution, and distillation is continued until all the ammonia is driven over. The distillate is collected in graduated glass cylinders after which it is treated with Nessler's reagent and the coloration produced matched with standard solution of ammonium chloride.

The standard solution is made up of a strength equal to 0.03815 gram ammonium chloride per liter so that one c.c. is equivalent to 0.01 milligram of nitrogen.

It is, of course absolutely necessary in determinations of this kind to take extreme care that the water, hydrochloric acid, etc., used are perfectly free from nitrogen compounds.

The table below shows the physical properties of these steels together with a column of merits obtained from the English formulas given above.

TABLE NO. 2—PHYSICAL PROPERTIES OF SAMPLES

Test Number	Elastic Limit, Pounds	Tensile Strength, Pounds	Elongation, Per Cent	Reduction of Area, Per Cent	Merit
1	68,000	105,500	18.5	46	65.5
2	58,400	98,400	19.5	52	63.5
3	69,500	105,000	21.5	42	68.5
4	70,400	108,900	17.5	41	66.5
5	53,700	95,200	17	34.5	60
6	82,000	123,500	19	45.5	74
7		Not	determined		

The steels containing titanium are especially given as the addition of this element is considered to retard the segregation of sulphur, phosphorus and carbon present; also, by its action in reducing a part of the nitrogen, it decreases brittleness and augments toughness and ductility. Its influence is said to be similar to manganese in reducing oxygen and in addition combines directly with nitrogen.

A lengthy paper on this subject by P. H. Dudley will be found in the *Journal* of Ind. and Eng. Chemistry, volume 3, pages 299 to 304.

In the heats referred to, the titanium was added as the steel was being poured into the ladle. In sample No. 2, 0.50 per cent, by weight, of ferro-titanium was added; this heat shows 0.080 per cent of titanium in the steel. Sample No. 3 about 0.30 per cent of titanium, by weight, was added in the form of rutile (red oxide of titanium ore); it will be observed in this case that the steel only shows a trace of titanium. It may be possible in this and, also, in sample No. 4, that the rutile added, being in fine powder and of light weight, floated on the surface of the molten steel and combined directly with the slag. In sample No. 4 about 0.60 per cent titanium was added in the form of rutile.

If we now apply Mr. Stromeyer's rule for quality judged from the phosphorus and nitrogen contents and assume the factor of P plus 5 (nitrogen) should not exceed 0.080 per cent, we get figures as follows:

Sample.	P + 5 (Nit.).	Merit.
1	.066	18.75
2	.053	33.75
3	.079	1.25
4	.068	15.00
5	.033	58.75
6	.038	52.50
7	Not determined.	

The merits obtained from the physical properties indicate sample No. 3 as being the next to the best steel, while from the P plus 5 (Nit.) formulae it is placed as last in rank, in fact it is very close to the danger point. This is only another case of the inconsistency or disagreement of merit figures. Sample No. 4 is third highest from its physical properties and is next to the last merit from its phosphorus and nitrogen contents. Sample No. 6, which is an extremely good steel as indicated by its physical properties, also excels in merit from its nitrogen and phosphorus contents; it will be observed that it is very low in both these elements.

The average of the open-hearth heats, sample No. 5, are extremely low in nitrogen; the physical properties are, also, good for this class of steel. These steels were made in a basic furnace and are far superior to the rail steel given as sample No. 7 if the elimination of nitrogen is a criterion. The functions leading to superiority of basic open-hearth steels are the possibilities for the elimination of sulphur and phosphorus and the reduction of the oxides. In this case extreme care in operation was taken; the quality of the steel produced shows the effects of this careful handling.

The writer trusts the merits and demerits of nitrogen in steel will be discussed further or until a definite and clear understanding of its effects on the quality of all grades of steel shall be firmly established. E. D. A.

The Western Metallurgical Field

Proposed Merger at Cripple Creek

Considerable interest attaches to the recent examination of the Golden Cycle mine in the Cripple Creek district by Mr. Richard A. Parker on behalf of the El Paso company. It is currently reported that Mr. Allen L. Burris, president of the El Paso, has been making an effort to consolidate some Cripple Creek mines, together with the district railroads, and that Golden Cycle mines and cyanide mill are part of the merger. A $25,000,000 holding corporation is part of the announced plan, and the properties to be acquired are the following: Golden Cycle mines and cyanide mill, El Paso mines, Colorado Springs & Cripple Creek District Railway, Midland Terminal Railway, Florence & Cripple Creek Railway, Gold Belt Railway, and the High and Low electric lines. The holding company will engage in general mining business, confining its activities to the Cripple Creek district for the present.

Copper Smelting in Michigan

Smelting practice in the Michigan copper district does not vary materially at the different smelters. The process is practically one of melting and refining, as the concentrates are mostly metal. At the Quincy smelter the concentrates are separated into three grades: No. O, mass, containing from 65 to 70 per cent copper; No. 1, coarse concentrate and some mortar discharge, averaging about 75 per cent copper; No. 2, fine concentrate, containing from 10 to 25 per cent copper. The poorest grade, No. 2, is agglomerated in a reverberatory furnace and later smelted in a blast furnace. This practice has superseded the former method of briquetting the fine copper for blast furnace reduction. No. 0 and 1 grades are melted in reverberatory furnaces having a capacity of about 50,000 lb. refined copper. The operation is intermittent, covering the following operations: The furnace is charged at 4 p. m., and the charge is skimmed every two hours after slag begins to form. This melting and skimming is continued for 15 hours, after which the charge is refined with air, oxidizing arsenic and other impurities, and forming some copper oxide. The molten bath at this stage contains about 95 per cent metal and 5 per cent oxide. The latter is then reduced by poling and with charcoal. Casting is commenced about 1 p. m. and finished about 3 p. m. The furnace is repaired if necessary and made ready for the next charge at 4 p. m.

Casting at the Quincy smelter is done by hand from ladles that are carried on an overhead track that runs above the line of molds. At the Michigan smelter a Walker casting wheel is in use, and at the Calumet & Hecla similar wheels are used in connection with two new reverberatory furnaces. The latter have capacities of 125,000 lb. of refined copper per day each, and are equipped with Stirling boilers for generating steam from waste heat, after the manner of similar equipment on reverberatory furnaces in the western copper smelters. There are some objections to the use of casting wheels; the cast

copper cools quickly, but if the wheel is moved before the surface is set, the ingots have a rough, wavy surface which is objectionable in subsequent rolling for industrial uses.

Reverberatory slags are high in copper, but if lower-grade slags were made it would be at the expense of the grade of metal produced; and as the slags would have to be resmelted in any event, there is no object in trying to get them as low as possible. Such slags will carry from 10 to 12 per cent copper, and are richer in metal toward the end of the skimming process. They are smelted in a low-pressure blast furnace, the product of which is copper of 95 per cent purity, which is later refined in a reverberatory. Slag runs continuously from the blast furnace, carrying about 0.8 per cent copper, which is about twice the grade of slags from western matte furnaces. Lower-grade slags involve reduction of iron which contaminates the copper. The principal flux used in the blast furnace is limestone, although iron in the form of pyrites cinder is used on occasion. Anthracite coal is preferred to coke, the latter being used when the furnace shows signs of running cold, or when accretions build up in the shaft.

Company Reports

The Shannon Copper Company has just issued a report covering the operations during the last four months of 1912. The company treated 95,235 tons of its own ore, producing 5,149,805 lb. copper, 1012 oz. gold and 54,394 oz. silver. The cost per lb. copper during the four months was as follows:

September	13.858 cents.
October	14.837 "
November	12.714 "
December	12.548 "

The high costs during this period were due to charging off certain increases in taxes and liability insurance which the legislature enacted last summer, and which could not be distributed over the costs for the year. Another cause contributing to high cost was the chilling of the large furnace during October, reducing production for several days without reducing expenses. The net profits for the period were as follows:

	Mines.	Railroad.	Total.
September	$41,000	$1,415	$42,415
October	29,900	1,039	30,939
November	66,200	1,850	68,050
December	65,695	1,773	67,468

The report states that the mines are in better condition than for years, and the lower levels hold out better promise than ever before. Dividends amounting to $300,000 were paid during the last eight months of the year.

The Nevada Consolidated Copper Company's report for the last quarter of 1912 shows a marked decrease in production compared with the preceding quarters, owing to labor difficulties which caused a complete suspension of operations from October 2 to October 24. The grade of ore milled, 1.44 per cent copper, was lower than usual. The production of copper was only 8,986,905 lb., less than half the usual production.

The reports of the *Utah Copper Company*, covering the fourth quarter of 1912, shows a monthly production of 4,302,194 lb. copper as compared with an average of twice that figure for the preceding months of the year. The decrease is ascribed to labor troubles, referred to in the last quarterly report, and to the shortage of labor caused by the emigration of foreign laborers for service in the Balkan War. The average assay of ore treated during the quarter was 1.104 per cent copper, as compared with 1.41 per cent for the preceding quarter. The average cost of copper, after making allowances for smelter deductions was 14.83 cents per pound, compared with 7.707 cents for the preceding quarter. The high cost was due to deficiency in tonnage, low grade of ore and extraordinary expenses. On the date of the report, Feb. 1, the mills were treating 18,000 tons per day, and it was expected that within thirty days the full capacity of 20,000 tons daily would again be reached.

The nineteenth annual report of the *Portland Gold Mining Company*, Cripple Creek district, Colorado, was issued on February 3, and shows the company to be in excellent condition, with greatly improved metallurgical facilities. The Colorado Springs mill has been reconstructed into an up-to-date cyanide mill, and has been operating at its full capacity of 400 tons per day since November 15, 1912. The new mill at Victor has continued to demonstrate its success in treating low grade ore which formerly was thrown on the dump as waste. Its capacity has been increased from 12,285 tons treated in December, 1911, to 14,791 tons in December, 1912. Additions have been made to increase the concentrating capacity, consisting of twelve concentrating tables. Further additions to plant consisted of four Pachuca tanks, one Dorr thickener and two classifiers. These changes have raised the original capacity of the mill from 300 tons per day to 500 tons. It is expected that the net profits from this mill will be equal to the dividend requirements of the company, leaving the profits from the old mill to be applied toward building up a substantial reserve fund. The average value per ton of ore treated in the new mill was $3.15, yielding a net profit of $1.17 per ton. The profits earned by this mill in 1911 and 1912 have equaled 90 per cent of its cost, and only a beginning has been made toward the treatment of the vast tonnage of low-grade ore available. The Cripple Creek drainage tunnel has lowered the water in the Portland mines 207 ft., and mining operations are now conducted on the sixteenth level, where the ore bodies are larger than above. All pending litigation has been compromised. The total net profits for the year were $325,509.84; dividends for the year, $240,000, and for the entire history of the company, $9,157,080.

Rochester, Nevada

The newest mining camp in Nevada which is attracting more than usual attention, is Rochester, in the Humboldt mountains, Humboldt county, being east of Oriana and Lovelock, which are on the Southern Pacific railroad. An incipient boom is now under way, and the prospect is that by late spring or early summer a new mining district will be well organized. New discoveries of high-grade silver ore have revived interest in a district which was active in the early sixties, when rich refractory ore was shipped from there to Swansea, Wales, for reduction. The ore now being mined resembles that of Tonopah, being high grade in silver and low in gold. At present two mines are shipping about 20 tons of ore daily to the Mason Valley smelter at Thompson, Nevada. A great deal of development work is being done, and some sales of property are being made. Approximately 2000 people are in camp, and more are coming in daily. The winter weather is reported to be favorable to prospecting. Owners of prospects for sale are much more conservative in their demands than they have been in other Nevada camps, and indications are that outside capital will be encouraged to come in. An element of permanency is found in the construction of a steam-electric power plant at Oriana, 10 miles distant, to transmit power to the mines. At present all mining is done by hand-drilling, and as the rock is extremely hard, little progress can be made.

Organizations of Mining Men

Matters of state and national legislation affecting the mining industry are resulting in the organization of mining men in different parts of the country. The question of mine taxation is the foremost issue in the different states, while the tariff on lead and zinc is the national issue around which much discussion is being waged.

The subject of taxation is before the legislatures of Utah and Colorado, and new organizations have been effected in both states to see that no unwise measures are passed. The principles of mine taxation unquestionably are not yet settled to the satisfaction of mine owners, nor is there any uniformity in the laws in different mining states. This state of affairs has received the attention of the American Mining Congress which, at its last session in Spokane, passed a resolution appointing a committee on mine taxation to report at the next session of the congress. In the meantime Colorado proposes a law that shall assess mining property at its full cash value, directing assessors

to take into consideration any matters that will enable them to arrive at a fair and equitable cash valuation. Manifestly this means nothing, and points the way to the great need of a scientific method of arriving at the value of mining property. Such a method as Colorado proposes will result in confusion and dissatisfaction. It has been remarked before by competent engineers that foreign countries have solved this question more satisfactorily than we have; that distinction is made between idle and active owners; that the former are subject to more than nominal taxation, and that output is taken as a basis for the taxation of active companies. It is to be hoped that our own states will accomplish something constructive and sound along this line, for many instances might be cited in which the question of unscientific taxation determines the difference between idleness and activity of some companies.

The proposed reductions of the tariff on lead and zinc are being vigorously opposed by the American Mining Congress under the active direction of the secretary, James F. Callbreath, who is at Washington. He has been well supported by representatives from Missouri, but assistance from other states has not been as vigorous as is desired. If the question has vital interest to producers of these metals, there seems to be a strange apathy among many of them.

Iron and Steel Market

The pressure upon steel mills for delivery of material showed no abatement in February. Demand against existing contracts continued strong, with specifications substantially equal to shipments, and efforts in many cases on the part of buyers to have shipments expedited against specifications already filed.

There have been no important advances or declines in the regular base prices of important finished steel products, and these prices are well maintained. There has been a very considerable reduction in the premiums paid for early delivery of certain products, also a diminution in the tonnage of material sold at premiums, so that the premium market is now relatively unimportant. Probably this change can properly be ascribed to the fact that we have been passing through the winter period.

In pig iron, which in recent years has presented a market altogether distinct from that of steel products, the slight weakening which occurred in January was carried farther. Pig iron prices no longer control steel prices, only a very small portion of the total merchant pig iron output being sold by direct barter with steel producers. Steel prices are affected only sentimentally, and then but very slightly.

Connellsville coke, which began to weaken late in January, slumped in the fore part of February, so that prompt furnace coke, which ruled at $4.00 a ton or higher from the beginning of November until late in January, sold at as low as $2.25, firming up slightly after considerable accumulations were liquidated, and making a market of $2.40 to $2.50 in the closing week of February. There being no important negotiations, a definite market for contract coke was not developed, but early in January it was impossible to contract for standard brands at less than about $3.50, while by the middle of February it became evident that if buyers and sellers got together it would hardly be on a basis outside the limits of $2.25 and $2.50. The slump in coke was due to several influences. The market had become unreasonably high, both in relation to cost of manufacture and in relation to selling prices of pig iron and steel products. There were fears in November and early December of an actual famine later on, production being usually curtailed around the holidays through too strenuous celebration by the workmen, and also being usually subject to curtailment through bad weather. Stocks were accumulated against these probable contingencies, but the winter proving exceptionally open production was not curtailed through natural conditions and in January furnaces began to use up stocks. At the same time more than half a dozen steel works furnaces began to work badly and took reduced coke shipments. Production increased slightly, and the combination of circumstances, increased production and reduced demand, although amounting to but a very small percentage of the total movement, was sufficient to change the complexion of the market completely. There is no permanent decrease in consumption, whereas with the freer supply furnaces will refuse to take the poor coke they willingly accepted in December and January, forcing restriction of output at the plants making poor coke, and this readjustment will probably hold coke at a moderately high level, though much below the unreasonably high level which recently prevailed.

Reports of business and financial activity outside the steel industry proper have been uniformly unfavorable, and it has been a surprise that the steel industry has not been affected. There are many who think it has been, evidently believing that unfavorable developments in the steel industry have been concealed. There is no reason to believe that their attitude is correct. The steel industry is in strong position, with a very large volume of sound business on books, and strictly new buying naturally light in consequence. The divergence between conditions in steel and in other branches of activity is explainable on the ground that the country has continued to develop steel requirements in recent years, while steel making and finishing capacity has increased very little in the past three or four years, in comparison with the rapid increase in demand in all previous time. This fact can readily be illustrated quantitatively.

In the decade ended with 1897 pig iron production increased 94 per cent. over the previous decade, and the next decade showed a further increase of 120 per cent., the average of these increases being 107 per cent. An increase of 107 per cent. over the 181,470,000 tons produced in the ten years ended with 1907 would require an output of 376,000,000 tons in the ten years to end with 1917. The five years of that period already elapsed have produced but 122,412,000 tons, leaving 254,000,000 tons for the five years 1913 to 1917 inclusive, whereas it is straining the industry's capacity at present to maintain a rate of a trifle over 33,000,000 tons. There are a few idle furnaces, but it would be impossible to swell the production rate to 35,000,000 tons a year and avoid loss to poorly positioned operations, unless finished products should advance more than raw materials and the alignment is such that this could not occur.

There is reason to conclude, therefore, that the steel industry remains active and in strong position when general business is not particularly good simply because, from a variety of influences, development of producing capacity has not kept pace with development of consuming capacity.

Pig Iron

Since its high point towards the close of December pig iron has declined an average of almost 50 cents a ton, the declines being practically uniform in foundry pig iron in all markets, but being very slight in steel making iron in the Pittsburgh valley district and in eastern Pennsylvania. The declines have occurred very quietly, in a dull and stagnant market, and it is likely that the development of a moderate demand would develop still lower, rather than higher prices. The practice of buyers of foundry iron is well known: on a rising market they buy farther and farther ahead, while given a weak market they refuse to buy until delivery has been completed on all their purchases. The furnaces are fully sold up for perhaps two or three months to come, and deliveries are being very well taken, so that iron is not accumulating, but it is fairly safe to predict that prices will continue soft until April or perhaps a later month, when the real test of the market will come. In the circumstances, it would not require an exceptionally heavy demand to stiffen prices again. The market now stands quotable as follows: Bessemer, $17.25; basic, $16.35; No. 2 foundry and malleable, $17; forge, $16.50, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh; No. 2X, delivered Philadelphia, $18.25; No. 2, foundry, f.o.b. Buffalo furnaces, $17; No. 2 foundry, f.o.b. Cleveland furnaces, $16.50 to $17; No. 2 foundry, f.o.b. Chicago furnaces, $17.25; No. 2 foundry, Birmingham, $13.50, with some resale iron offered at $13.

Steel

Never, perhaps, have transactions in billets and sheet bars been so light. There has been practically no steel available for early shipment, and for such later shipment as the mills could compass they simply refuse to quote. As representing the general temper of the market we quote billets at $29, sheet bars at $30 and rods at $30, f.o.b. maker's mill, Pittsburgh or Youngstown.

Finished Steel

Effective February 1, the National Tube Company advanced steel boiler tubes one point, or about $2 a ton, other makers immediately concurring.

Regular prices at Pittsburgh, unless otherwise stated, are given below, subject in some cases to premiums for early delivery:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.50 cents.

Steel bars, 1.40 cents.

Steel hoops, 1.60 cents.

Common iron bars, 1.65 cents, Pittsburgh; 1.65 cents, Philadelphia; 1.60 cents, Chicago.

Sheets, blue annealed, 10 gage, 1.75 cents; black, 28 gage, 2.35 cents; galvanized, 28 gage, 3.50 cents; painted corrugated, 28 gage, 2.55 cents; galvanized corrugated, 3.55 cents.

Tin plates, $3.60 for 100-pound cokes.

Merchant steel pipe, 3/4 to 3-in., 80 per cent. off list.

Steel boiler tubes, 3 1/2 to 4 1/2-in., 70 per cent. off list.

Standard railroad spikes, 1.85 to 1.90 cents.

Button head structural rivets, 2.20 cents; cone head boiler rivets, 2.30 cents.

The price of 1.45 cents now quoted on plates and shapes is the price which the steel corporation has regularly maintained; for a time independent manufacturers endeavored to obtain 1.50 cents and as the steel corporation accepted only selected business at its quotation the market was quoted in general at 1.50 cents. As it is now recognized that the independents can book little attractive business at higher than 1.45 cents the general market is now quoted at that figure.

John Fritz

In John Fritz the engineering profession and the steel industry has lost a master mind, the United States one of its greatest citizens. A self-made man and a self-taught engineer he was a peer of the most eminent steel men of three generations and the engineering societies honored themselves when on the occasion of his 80th birthday they established the John Fritz medal to perpetuate the memory of his life work.

In view of the fact that he left for us to read his delightfully interesting autobiography there is no necessity here to go into details. Only a few dates and a few attainments shall be mentioned.

John Fritz was born Aug. 21, 1822, in Londonderry Township, Chester County, Pa. His early days were spent on a farm. At the age of fifteen he left school and became a blacksmith's apprentice. He became interested in foundry work, obtained employment with the Norristown Iron Works in 1845, and within a few months was superintendent of the plant. In 1849 he left this position and accepted one as mechanical engineer to put up the machinery for a rolling mill at Safe Harbor, Pa.

His health soon failed at Safe Harbor, and he was forced to leave. Then for several years he struggled with fever, working only a part of the time. His services were much sought, both at Safe Harbor and at Norristown.

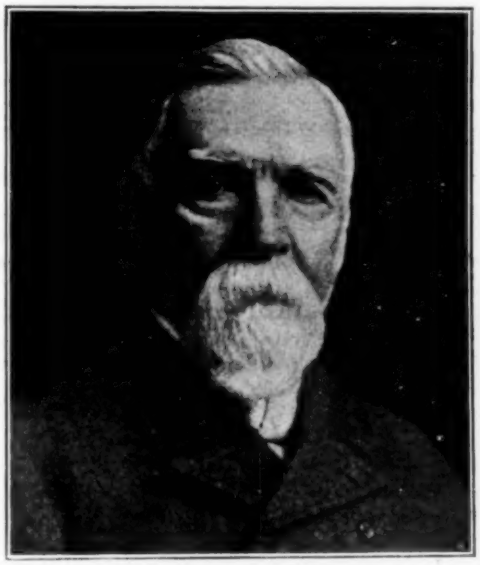
In 1854 one of his former employers became interested in the Cambria Iron Works, at Johnstown, Pa., and induced Mr. Fritz to go there as general superintendent.

One of his innovations there was the replacing of the two-high rolls with three-high rolls, to effect greater economy and lessen the likelihood of accidents. This was made in 1857. In 1860 he left the Cambria Iron Works and became general superintendent and chief engineer of the Bethlehem Iron Company, which was then about to build a plant at Bethlehem, Pa.

In 1864, the Bessemer process was introduced into the United States. Mr. Fritz had been studying the matter of steel-headed rails, and although the first steel rails in this country were failures, he induced his employers to build a Bessemer plant, believing the failures were due to an unnecessary amount of phosphorus. The plant was started in 1868. About the same time the manufacture of acid open-hearth steel began, using the Siemens regenerating furnace. Three years later Mr. Fritz had a Siemens furnace installed at the Bethlehem plant, with important modifications of his own that proved successful.

The following year he added a blooming mill. He then turned his attention to the making of plates and structural shapes, but his company would not take up their manufacture. He later succeeded, however, in getting the company to install a forge and armor-plate plant, although there was at that time neither the demand for the thick steel plate nor proof that it could succeed. It did succeed in tests, and the demand was created.

Mr. Fritz retired from active work several years ago. Previous to his death he had been in feeble health for more than a year. He died at his home in Bethlehem on February 13.



THE LATE JOHN FRITZ

The funeral services were held at his late residence, which, though spacious, could not contain his many hundreds of friends, including eminent men of all callings and the delegations from the national societies. A special car brought engineers from New York, and there were present a large number of Lehigh University students. Thousands of the townspeople and steel plant employees waited patiently in the winter wind to follow "Uncle John Fritz" reverently a few hundred feet to his last resting place above the valley where the vast steel plant which is one of his enduring monuments extended for miles in unwonted silence.

George W. Maynard

George W. Maynard, prominent as mining engineer and metallurgist and highly esteemed as a man by all who had the pleasure to know him, died in Boston on February 13, after an illness of several weeks which had started while he was on a professional mining trip to the West.

Mr. Maynard was born in Brooklyn, N. Y., on June 12, 1839. He graduated from Columbia as one of the famous class of 1860—the first class of the department of Chemistry at Columbia.

Last September (Vol. X, p. 517, 1912) it was the privilege of this journal to publish from Mr. Maynard's own pen rem-

iniscences of those earliest days of chemistry in New York City, together with the picture of the class of 1860 of the department of chemistry. Mr. Maynard wrote these reminiscences just before he started on his last trip to the West. They were his last contribution to literature and were characteristic of the man and his true big heart.

In 1860 he went to Germany and for a year studied chemistry, physics, and mineralogy in the University of Goettingen, under Woehler. While there he was a fellow student with J. Pierpont Morgan. Leaving Goettingen he spent a year at Clausthal in the Hartz Mountains with Bruno Kerl, studying mining and metallurgy in the School of Mines there. In 1862 he received the degree of Master of Arts from Columbia.

Mr. Maynard's professional practice started with an engagement at the Connoree mines in County Wicklow, Ireland, where he remained a year occupied in chemical and metallurgical work. In 1864 he returned to New York to establish an engineering office and chemical laboratory, and in the spring of the same year he made his first professional examination of a mining property in Colorado. He liked the West and in December, 1864, he returned to Colorado to establish an engineering office and assay laboratory in Gilpin County.

Mr. Maynard returned to New York in the winter of 1867, taking charge of a sulphuric-acid plant on Staten Island. In 1868 he was appointed Professor of mining and metallurgy at the Rensselaer Polytechnic Institute in Troy, N. Y., and occupied that chair until 1872. From 1873 to 1879 he was in Europe. His headquarters were in London, but during this time he spent six months in eastern Russia, erecting a copper reduction works for a British company at Voskrevsenky.

His stay in London became important for his future, as he was consulting engineer for several iron and steel works in England and Wales, and in such capacity made at Middlesbrough, England, the first demonstration in a large way of the basic Bessemer process of steel making. He was a very intimate friend of Sydney Gilchrist Thomas.

Mr. Maynard returned to New York in 1879 for the purpose of introducing the Thomas-Gilchrist process into the United States, and sold the patents to the Bessemer company, afterward known as the Steel Patents Company. Subsequent to that time Mr. Maynard always resided in this country, though he made various brief professional trips abroad.

Mr. Maynard continued his consulting practice until the time of his death. He died in the saddle. His many friends will remember him as a mining and metallurgical engineer of high attainments and as a man of lovable character.

Raphael H. Wolff

Raphael H. Wolff, widely known through his long connection with the iron and steel industries of both the United States and Germany and prominent through his great activity in introducing the electric steel furnace in this country, died on January 23 in a sanitarium in the Grunewald, near Berlin, Germany. Mr. Wolff had gone to Europe last June on one of his trips in which he used to combine rest and recreation with study of commercial and industrial conditions and none of the many friends who saw him off on that fine Saturday morning on the dock in Hoboken, could have imagined that it was a farewell forever.

Mr. Wolff came of an old family of wire manufacturers, with a mill at Neheim on the Ruhr in the Westphalian district. He came to the United States in 1870 and first found employment with the Pratt and Whitney Co., of Hartford.

Mr. Wolff was then for many years an importer of iron and steel in New York City, chiefly of wire rods and steel billets. Later he established a wire mill at 117th Street and the Harlem River, New York City, and conducted it as the head of the firm of R. H. Wolff & Company, Ltd. There he manufactured for many years cold rolled strip steel and high-grade wires, including piano wire and watch-spring steel ribbon. During the period of the bicycle craze he also took up the building of wheels.

Mr. Wolff sold his plant in 1902 to the Washburn Wire Company. He then became connected with the Crucible Steel Company of America. He became their European representative and established their European department with headquarters at Hamburg, Germany, and was soon successful in building up an important business.

Some ten years ago Mr. Wolff investigated for an American company the Heroult electric steel refining process and reported favorably on it, but his recommendations were not followed. It is a pity on account of the private character of this report that extracts from it cannot be given here, as that report was a model of business sagacity, proving that Mr. Wolff was one of the earliest practical iron and steel men who grasped the potentialities of electric refining.

As a result of his firm conviction in the great feature of the electric steel furnace Mr. Wolff became connected with Dr. Paul Heroult as his American representative and as such he developed an untiring activity in interesting the big men in the American iron and steel industry in the electric furnace. By word and by writing he conducted a persistent campaign for the Heroult electric steel furnace. His articles on the use of electric furnace for making alloy steels, on commercial electric steel and gas power and the possibilities of the electric furnace in steel foundry practice, on the electric furnace for rail and ordnance steel, published in this journal, volume 6, pages 6, 225, and 485 (1908) will even now repay rereading. He was one of the earliest men to recognize the importance of dividing properly the metallurgical reactions between the ordinary oxidizing furnace and the electric refining furnace so as to reduce the cost of electric refining. He took out a patent for a process the chief feature of which was to restrict the treatment in the electric furnace simply to the object of finishing and resting the metal so that any gas or slag left in the metal should rise to the top (this journal, volume 8, page 288, 1910).

THE LATE R. H. WOLFF

Mr. Wolff conducted the negotiations which led to the installation of various Heroult electric furnaces in American steel plants and finally resulted in the acquisition of the American rights in the Heroult furnace by the United States Steel Corporation.

Mr. Wolff was also largely instrumental in the establishment of the ferro-silicon plant of Electro-Metals, Ltd., at Welland, Ontario. He was the president of this company for several years and was on its Board of Directors until the time of his death.

Mr. Wolff was a man of strictest business integrity, a man of strong character and a kind heart. His untimely death is mourned by many friends.

Pulverized Coal as a Fuel

By H. R. Barnhurst

Coal properly pulverized and burned may be made to yield higher economic results than are attainable by any other means.

The requirements necessary to success, while simple, are absolute and must be obeyed.

First—The coal must be dried so that it contains not over 1 per cent of moisture.

Second—The coal must be pulverized to a high degree of fineness.

Third—It must be projected into a chamber hot enough to cause instant deflagration.

Fourth—It must be supplied with air sufficient to yield the oxygen necessary to burn the carbon of the coal at once to CO_2.

Taking up these requirements in order, the drying of the coal to a moisture content of not over 1 per cent is indispensable. Coal does not grind well if moisture in excess of this be present.

In burning coal the moisture, free or combined, must be disposed of either in the process of preparation or in the moment of combustion. In the latter case not only is the efficiency of the furnace lowered by the calorific investment in the superheated steam passing out as a product, but the temperature of the furnace is lowered materially. The drying of wet coal in the furnace itself, is doing this necessary part of the work in the most expensive place and at the cost of temperatures which may be essential to the industrial process of which high heat is a factor.

Fine grinding—With the best type of machines obtainable for this purpose, the coal and its contained impurities may readily be powdered to such a degree that under the screen tests 85 to 90 per cent will pass through apertures 1/400 in. square, while the total residuum left upon a screen whose apertures are 1/200 in. square, will be from 2½ to 5 per cent and this residuum would pass through screens of 1/100 in. square. It must, however, be borne in mind that of the percentage passing the smaller apertures 1/400 in. square there is a high percentage of absolute dust or impalpable powder not commercially measurable. This is proven by the fact that in tests made upon calibrated screens of 1/600 in. square apertures, over 70 per cent still passed through. It certainly appears to be safe to assume, therefore, that the average size of the particles would be below a cube measuring 1/600 in. on the side.

It may be interesting, therefore, to state that the total numbers of particles resulting from the powdering of 1 cubic inch of coal to the dimensions given would yield 216,000,000 grains of dust. Simple calculation on this basis shows that while a cubic inch of coal exposes 6 square inches for the absorption and liberation of heat, the surface exposed for the same purposes by the powdered coal is 25 square feet. Inasmuch as no fuel burns until it is heated to a temperature at which it develops more heat than it receives, the advantage of this enormous absorbing and delivering surface is apparent. The result of this is shown in the clearness and uniformity of the flame produced. Where coarse particles are permitted to enter the furnace, the distinct sparkles are apparent. These larger particles are carried beyond the region of oxygen supply and are for this reason not fully burned.

Third—While coal ignites freely, in a hot chamber, this ignition means the absorption of heat from somewhere, and if the coal rapidly projected by air does not develop its heat near the point of ignition, means must be devised to maintain the heat necessary for ignition where it is needed, i. e. at the first entrance of the coal into the furnace. It is apparent, therefore, that giving the fuel too great velocity upon entrance is not good practice.

Considering the fourth requirement along with the third we would say that some singular errors and misconceptions have attended the practices of many users of powdered coal. More particularly do we refer to the use of large fans to supply the air necessary for the projection of the fuel, where the air nozzle has been reduced from 16 in. or 18 in. diameter to 4 or 5 in. at the jet under the expectation that all of the air in the 16 in. or 18 in. pipe would be hurried through the 4 or 5 in. nozzle if not a smaller one. The futility of this is apparent.

To describe the operation more clearly, the coal is received in a bin over the feeders. (Fig. 1.) Its weight is about 38 lb. per cubic foot when loose in the bin. Settling awhile brings the weight to about 45 lb. per cubic foot by displacing the entrained air. Across the bottom of this bin and within a pipe extending horizontally from it is a double-flight worm or feed screw. This double-flight screw resists the tendency of the light coal to flow of itself along the feed pipe. This screw extends over a flanged pipe-cross into which the fuel is delivered. The rear end of the screw is supported by a bearing in a flange on the side of the bin near the bottom, the shaft projecting to receive a driving pulley or chain sprocket. The delivery end of the screw shaft is supported by a bearing in the cover of the horizontal opening of the flanged pipe-cross. The top opening of the cross is uncovered to permit the air

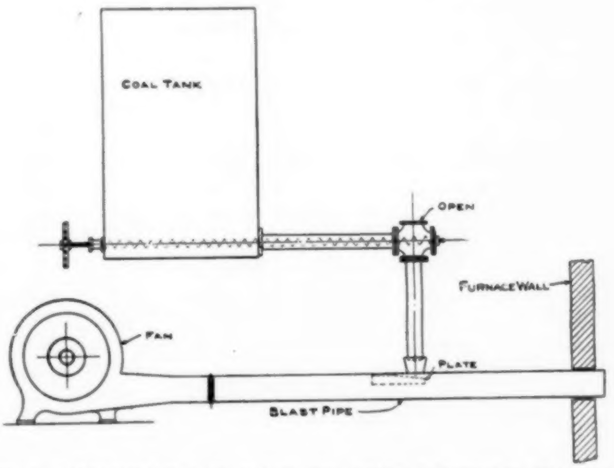


FIG. 1.—ARRANGEMENT OF PLANT FOR USING PULVERIZED COAL AS FUEL

to draw down with the falling fuel. This fuel dispersed in the air so drawn in, descends a vertical pipe attached to the lower opening of the cross, the pipe being long enough to be within the funnel or injection pipe. At the bottom of the funnel is a diagonal plate upon which the fuel falls. The plate is tight against the air pipe up the current and flared open on the side, towards the furnace down the current and takes up about one-fourth the diameter of the pipe. This forms at this point a "vena contracta" producing a suction in the funnel, drawing in through it, supplementary air with the fuel. The fuel spraying upon this plate mixes very thoroughly with the air from the fan, the eddy currents caused by it, assisting very materially its dispersal through the main column of air supplied by the fan.

The admission funnel should be far enough from the furnace to permit this mixture to be thorough. Too high pressures defeat this somewhat, as well as tending to project the fuel too far into the furnace before flashing. As soon as this fuel cloud begins to absorb the heat of the chamber into which it passes, a rapid expansion of the air takes place, separating the particles of fuel in suspension, in the proportions of the absolute temperatures to the temperature of the initial air. It is a matter of discussion whether the best results are obtainable by a delivery of all the air found necessary for combustion by the feed pipe together with the percentage of excess air found to produce the best results, or to use a smaller quantity of air in the feed pipe and look for the further supply from other openings.

Good practice would seem to point to absolute control of

air by the fan and its gates, and the fuel by the varied speed of the feed screw. The furnace should have a good natural draft to a chimney controlled by a damper. It must be remembered that perfect combustion of one pound of carbon demands 2 2/3 pounds of oxygen. This is contained in 11.6 pounds of air or about 154 cubic feet; should less than this be supplied a proportionate amount of fuel will be burned to CO with a loss of two-thirds of its initial efficiency, a part of which may be regained by contact with heated oxygen, or it may pass on and burn in the chimney, doing no good if the heated oxygen is brought there in first contact with it. If the oxygen is not heated sufficiently smoke will result and the full heat be undeveloped.

The greater the volatile constituents of the coal the more readily will it deflagrate, as these gases distil from the fuel and ignite at a temperature lower than that required for the carbon itself. Their need for oxygen is, however, greater per pound of fuel (or gases) than that of carbon, and is proportional to the heat evolved. Their average value is in heat units nearly 50 per cent more than that of carbon.

The temperatures attainable with pulverized coal are very high, so high that excess air is commonly admitted in proportions ranging between 50 and 100 per cent. This will be shown by the following table based upon the perfect theoretical combustion of carbon with proportion of air given:

1 lb. carbon with 11.6 lb. air	......Normal.	4859° F.
1 lb. carbon with 12.76 lb. air	......10 per cent excess.	4448° F.
1 lb. carbon with 13.92 lb. air	......20 per cent excess.	4102° F.
1 lb. carbon with 15.08 lb. air	......30 per cent excess.	3807° F.
1 lb. carbon with 16.24 lb. air	......40 per cent excess.	3550° F.
1 lb. carbon with 17.40 lb. air	......50 per cent excess.	3326° F
1 lb. carbon with 18.56 lb. air	......60 per cent excess.	3129° F.
1 lb. carbon with 19.72 lb. air	......70 per cent excess.	2954° F.
1 lb. carbon with 20.88 lb. air	......80 per cent excess.	2797° F.
1 lb. carbon with 22.04 lb. air	......90 per cent excess.	2656° F.
1 lb. carbon with 23.20 lb. air	.....100 per cent excess.	2529° F.

In practice the furnace tender speedily becomes educated to the point of judging whether a fire is hot enough by its color and by the length of the flame. The more perfect the conditions the shorter and whiter the flame will be.

The pulverized coal introduces very effectively the element of time into the equation. Given a pound of fuel with say 15,000 heat units, these may all be developed by slow combustion at low temperature, or by burning the fuel in pulverized form, quick combustion gives high temperature. The same quantity of heat developed in both cases, but in one instance in a minute and in the other half an hour.

The influence of preheated air upon the economy of the burning of any fuel resolves itself into ascertaining the quantity of fuel which would be necessary to bring the air to the preheated temperature plus the heating of the excess air also to that temperature. Except as a means of transporting heat excess air has no effect in furnace as far as the fuel combustion is affected. This preheating to be of any economical value must be obtained from heat which would else be wasted. To heat all the primary air necessary to combustion and 50 per cent excess air to a temperature of 1000 degrees in excess of the surrounding temperatures would show a saving of some 4100 heat units or nearly 30 per cent of the fuel value. But few of the industries, however, outside of the metallurgical arts permit the waste heat to pass off at such temperatures and volume as to be available. The regenerative checker-work of open-hearth steel furnaces is the best example of success in this preheating.

In this case it has been a necessity to boost the temperature in this way, because the gases from gas producers burned cold, would not give the temperatures necessary for the work to be done.

We may say that four heat units per degree would represent the saving achieved per pound of fuel fired with 50 per cent excess air by heating all the air admitted to the furnace. The measure of efficiency is dependent upon the loss finally carried away in the rejected gases. In the case of regenerative furnaces this may be lower than in furnaces not equipped with regenerating checker-work, but if to the percentage of loss with regeneratives be added the losses in the gas producers, the pulverized coal directly fired will afford the greater economy of operation.

With the means at hand of obtaining quickly and safely heat of greater intensity than by the use of coal upon grates or in producers, it would appear to render practicable further steps in many of the arts hitherto restricted by the limitations of furnaces at command.

The fear of explosion of coal need not enter into consideration. Coal lying in a bin or conveyor does not explode. It is only when mixed with air or supported by air currents that coal will "puff." In burning it, therefore, we do not mix the coal and air until just as it enters the furnace at high velocity. Against this column of inrushing air and coal the puff cannot take place.

The air is introduced before the coal is turned on, and the coal is shut off before the air. Only by introducing coal faster than it can burn will an explosion occur and then the effect is trifling. It is the gas produced and not the coal that causes this. It wants oxygen and comes outside to get it.

The presence of impurities in the fuel has not much effect. Of course, only combustibles will burn; the incombustibles are inert and do not affect the operation of the furnace. Their effect is in the lessened results from a dollar's worth of fuel negatived by a goodly percentage of waste substance. The writer has burned effectively fuel in which analysis showed 52 per cent of ash. Let us reiterate the conditions—*Dry coal, fine grinding, hot chamber or fire box, proper air supply.*

An important part of the subject must not be overlooked. The durability of the furnace is, of course, vitally essential. In the metallurgical arts when extreme heat is an essential part of the operation, care must be taken to avoid destroying the furnace by its own operation. This is not difficult. Much of the troubles have come from the gases impinging upon the furnace walls at points where change of direction of gas travel is necessary, and from too high velocity of gases due to contracted ports.

If the utilized heat is largely absorbed from the gases by the charge, the waste gases will be proportionately less active in scouring the brickwork. In almost any construction except perhaps a rotary kiln it is found necessary to change the direction of the gases in their progress toward the flue. This change of direction causes the gases to impinge upon the diverting bricks with an energy proportional to their velocity. The brick at these points can be fully protected by a system of water-cooled pipes embedded in the walls. The brick may frit somewhat until the area of protection is reached, when further progress is arrested.

The surprisingly small amount of water which it has been found necessary to introduce, maintaining the outlet below 200° F. proves that the cooling effect is limited to a prevention of cumulative action and is not perceptibly a drawback upon efficiency. Of course, the piping must be so arranged that no air or steam pockets shall exist and that the circulation will be proportional to the heat stimulus. One other point and we will conclude. The pulverized coal furnace has no ups and downs. There is no thick fire or thin fire, fresh coal or old coal to insure fluctuations.

The furnace can be always kept at its best working point and so kept it will be heated equally all over. Of course, a large charge of metal to be heated will by its very volume absorb heat rapidly causing a fall in waste gas temperature and possibly a little smoke at first. This is in the nature of things, but the extremely effective conditions quickly bring the charge to a point where the chill is not sufficient to affect combustion and high temperatures come again and smoke disappears. If the work to be done is constant, there is no reason why high conditions may not be uniformly maintained by proper construction and operation.

We believe the subject has been mastered to a point beyond the experimental stage where the full benefits of high efficiency

may be confidently relied upon in this beautiful method of burning coal. As before mentioned, the quality of the coal is not with this method of supreme importance. Indeed, its great value in the developments of the future may lie in the efficiencies obtainable from low-class or refractory fuels hitherto unavailable.

Fuller Engineering Co., Allentown, Pa.

An Investigation of the Air Lift Pump

The air lift method of pumping, though not highly efficient as compared with some other methods, is nevertheless an important one, owing to the many advantages it possesses over other methods in pumping corrosive liquids and raising large quantities of water from wells of small bore. Notwithstanding the fact that this method of pumping has been known for over a century and is now quite extensively used, the amount of reliable data available to the practicing engineer concerning the performance of this type of pump is very meager.

With the purpose of supplying the demand for reliable data, extensive experiments were undertaken in the hydraulic laboratory of the University of Wisconsin under the direction of Messrs. **George J. Davis** and **Carl R. Weidner,** assistant professor and instructor, respectively, in hydraulic engineering in the university. The results of the investigation have been published in *Bulletin* No. 450, Engineering Series, Vol. 6, No. 7, University of Wisconsin, which may be obtained by remitting 40 cents to the Secretary of the Regents, Madison, Wis.

The *Bulletin* gives interesting information on the history and development of the air lift pump, showing that the principle was first used by Carl Emanuel Löscher, a German mining engineer, in 1797. The principle of the air lift is discussed and the various theories of its operation are given. The four types of air lift pumps are illustrated and described, namely, the side inlet, the annular air tube, the central air tube and the combination (annular and central air tube).

Among the disadvantages of the air lift pump are cited its low hydraulic efficiency, 25 per cent to 33 per cent; the great depth of submergence required; its limited efficiency in horizontal pumping, and the aeration, which in some circumstances is a disadvantage, causing rusting of the eduction pipe and the deposition of salts in foot-piece.

The principal advantages lie in its large capacity, low maintenance cost, low operating cost, immunity from effects of high temperatures in the liquid pumped, and its reliability.

The investigations at Wisconsin are described in detail, and the results of other experiments are also given. The variables which affect a particular size and type of pump are the following: (1) percentage of submergence; (2) lift; (3) discharge; (4) volume of air; (5) pressure of air. The conclusions which may be justifiably deduced from the Wisconsin experiments are given below, and hold only for the particular size, type and length of pump on which the experiments were performed. The inference, however, may be drawn that these conclusions would hold for other types and sizes.

(1) The central air tube pump has the greatest theoretical capacity for a given size of well.

(2) The coefficient of pipe friction and slip decreases as the discharge increases, and decreases as the ratio of volume of air to volume of water increases.

(3) The coefficient of pipe friction and slip varies with the length of the pump, but seems to be independent of the percentage of submergence and of the lift.

(4) The length of pump, the percentage of submergence, and therefore the lift, remaining constant, there is a definite quantity of air causing the maximum discharge. This quantity of air for maximum discharge, as also the ratio of volume of air to volume of water, differs for different percentages of submergence and lift, the length of pump remaining constant.

(5) The length of pump remaining constant, the maximum output, *e.g.*, foot-gallons, occurs at about the same percentage of submergence for all rates of air consumption, being from 61 per cent to 65 per cent for the pump used in the Wisconsin experiments. At other submergences the output varies as the ordinates of a parabola having a vertical axis. Under these conditions the lift does not remain constant as the percentage of submergence varies.

(6) The length of pump and percentage of submergence remaining constant, and therefore constant lift, the efficiency increases as the input decreases, that is, the higher efficiencies are obtained at the lowest rates of pumping.

(7) By varying the percentage of submergence, and therefore the lift, the length of pump remaining constant, the maximum efficiency is obtained at approximately 63 per cent submergence for all rates of input or discharge.

(8) The lift remaining constant, the efficiency increases as the percentage of submergence increases, for all rates of input and all practical percentages of submergence.

(9) With the same size and type of pump, the percentage of submergence remaining constant, the efficiency increased as the lift increased for the small lifts experimented on, that is, up to about 24 ft. From a theoretical study, however, the indications are that a point will be reached from which the efficiency will decrease as the lift increases.

(10) Other conditions remaining constant, there is no advantage to be gained by introducing compressed air above the surface of the water in the well.

(11) The type of the foot-piece has very little effect on the efficiency of the pump, so long as the air is introduced in an efficient manner and the full cross-sectional area of the eduction pipe is realized for the passage of liquid. Anything in the shape of a nozzle for increasing the kinetic energy of the air is detrimental.

(12) A diverging outlet which will conserve the kinetic energy of the velocity head increases the efficiency.

A Test of an Edison Primary Battery (Edison-Lalande Cell)

By Dudley Sanford

A precision to within 1 per cent is sought in this experiment.

The battery tested was a new Edison primary battery, type "RR," charged according to the directions inclosed with the cell. All battery connections were soldered to avoid contact resistances.

The test was made by the well-known condenser method. This method involves the determinations of the p. d. across the terminals of the cell when it is on open circuit and when it is closed through a fixed resistance. To obtain these a condenser is charged to the unknown potential and is immediately discharged through a ballistic galvanometer, whose constant has been previously determined by charging the condenser to the potential of a standard cell and allowing it to discharge through the galvanometer. $d = kCE$, where d is the deflection using the standard cell, C the capacity of the condenser, E the e.m.f. of the cell, and k is a constant. Using another cell, $d' = kCE'$. Consequently

$$E' = \frac{d'}{kC} = \frac{d'E}{d}$$

The apparatus consisted of a 1.5-microfarad condenser, an external resistance of 0.126 ohm (from terminal to terminal of the cell), a D'Arsonval galvanometer with telescope and scale, and a charge and discharge key (K in diagram).

The procedure followed was this:

At time 0 min. open circuit reading taken with key K' open,
At time 1 min. closed circuit reading taken with key K' closed.
At time 2 min. open circuit reading taken with key K' open,
At time 3 min. closed circuit reading taken with key K' closed,
etc., to end of 60 min. K' was then opened permanently, and readings were taken at intervals of 2 minutes. The internal resistance $x = R(E - E')/E'$, and the current $I = E'/R$, where E represents open-circuit voltages, E' closed-circuit voltages, R the external resistance in ohms, and I the current in amperes. The curves showing these quantities are given.

There are several interesting facts to be noted from these curves. The cell has an abnormally high initial e.m.f., 1.03 volts, which drops very rapidly for a short time and is then practically constant at 0.7 volt, though decreasing slightly. The p. d. at the terminals, however, increases gradually at first, but at the end of 60 minutes is nearly constant at 0.55 volt. The current is seen to rise rapidly at first and even at the end of the test is still rising, though not so rapidly. This is explained upon observing the internal resistance curve, for this resistance falls as the cell is used and is at all times low, being but 0.037 ohm at the end of the test. The cell recovers its normal voltage in a comparatively short time.

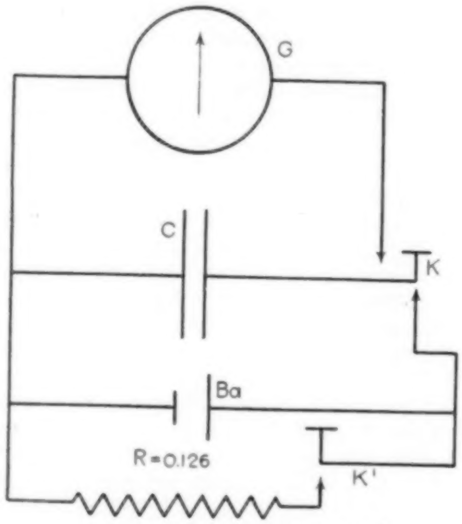


FIG. 1.—ARRANGEMENT OF TEST

After running the above test the battery was placed in series with an ammeter and a copper voltameter and allowed to run until completely exhausted. A voltmeter with a key in circuit was placed across the battery terminals. Readings were taken on the two instruments at suitable intervals of time, which ranged from two hours to thirty hours. The voltmeter readings were correct for the IR drop due to the internal resistance of the cell. From these readings and the proper weighings these results were obtained:

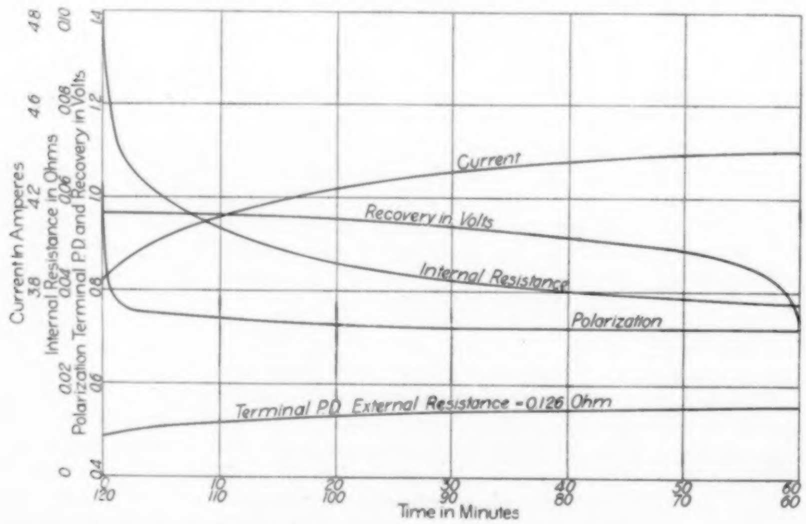


FIG. 2.—TESTS OF EDISON-LALANDE CELL

According to the readings of the instruments the total amp-hours (excluding first test) was 389.3. By weighing it was found that 466.05 g. of copper had been deposited, and this corresponds to 392.88 amp-hours. This would show that the readings were correct to 0.9 of 1 per cent. An accuracy to 1 per cent is therefore assumed in the other data obtained. The zinc consumed was 472 g., or 1.2 g. per amp-hour, or 0.000333 g. per coulomb. The accepted value for the electrochemical equivalent of zinc is 0.0003387 g. per coulomb, a difference of only 1.4 per cent.

But, in general, the value of the cell will depend not only upon the quantity of electricity which it will give, but also upon the *work* it is capable of doing. To know the capacity of a cell only in amp-hours gives inadequate information on which to determine its value to the user. But multiplying the volts by the amperes and integrating over the time of the test gives the watt-hours, a very important factor to be known. This particular cell gave 181 watt-hours of work. Some of this is always lost internally, and the efficiency of a cell varies with the constants of the circuit of which it is a part. But taking the makers' rate of maximum continuous output, 6 amp, and the voltage of 0.7 as given by the test, we find that the efficiency would be 79 per cent. The useful work obtainable would then have been 143 watt-hours. This, however, is the lowest efficiency for continuous output.

From the weighings of the zinc we find that 3.3 g. were consumed per watt-hour, or 3300 g. per kw-hour. The manufacturers' list price for the zinc plates is $0.00091 per gram. Consequently the cost of producing energy at the lowest efficiency is $3 per kw-hour. At 100 per cent efficiency it would be $2.37 per kw-hour. This is on the assumption that the zinc is the only substance consumed in the production of available energy and that the other materials will be recovered. If, however, the user will throw away the remaining materials and buy a complete renewal the cost will be $11.90 and $9.40 per kw-hour at the two efficiencies. If in either case the materials are bought in large quantities this cost may be much reduced—50 per cent in lots of $2,000 or more and proportionately for intermediate quantities. We thus note that under favorable conditions electrical energy obtained from this highly efficient type of primary battery costs around fifty times the ordinary market price of electrical energy.

If the total energy of the cell externally available at its lower efficiency were applied to raising a kilogram weight it would raise it 52,500 meters, or a ton weight a distance of 192 ft. At its full efficiency it would raise a kilogram weight 66,400 meters or a ton weight 240.3 ft.—a total energy of nearly 250 ton-feet. The total weight of the cell was about 20 lb.

The manufacturers of this cell now have a cell (BSCO No. 401) which they claim, on the basis of their tests, is even more efficient than this, yielding 247 watt-hours, or 1.36 times the energy of the type "RR" cell, but its cost for a complete renewal is 1.24 times that of the "RR" cell. Consequently from the cost standpoint it is 10 per cent more efficient than the "RR" cell. The action of the cell, however, is practically the same as is shown in the curves of the first part of this experiment.

NOTE:—The above test was made under my immediate observation and I believe the results are reliable.—EDWIN F. NORTHRUP.
Palmer Physical Laboratory,
Princeton, N. J.

The Data of Geochemistry is the title of Bulletin No. 491, issued by the U. S. Geological Survey, which will be sent free on application. It comprises a manual of geologic chemistry, including chapters on the nature distribution and relative abundance of the chemical elements, the composition of the atmosphere and of volcanic gases and sublimates, the mineral content of surface and underground waters, the nature of saline residues, the molten magma of the earth's interior, the rock-forming minerals, the composition of igneous, sedimentary and metamorphic rocks, rock metamorphism and decomposition, metallic ores, natural hydrocarbons, coal, lignite and peat. The Bulletin is exceedingly interesting from more than one viewpoint.

Minerals Separation Flotation Plant at Kyloe Copper Mines, N. L.*

By H. Hardy Smith

The ore treatment is almost pure copper pyrites (chalcopyrite) in a gangue of quartz and quartz-felsite. Very little free iron sulphide (pyrite) occurs, and at the time of the erection of the flotation plant the stopes in the oxidized zone had been almost entirely depleted.

This zone only extends to a shallow depth, and, below the 100-ft. level, the sulphide for the most part is clean and bright; but in the crushed and brecciated parts of the ore channel, where surface water has circulated freely, the gangue is decomposed and kaolinized, and the copper sulphide badly tarnished.

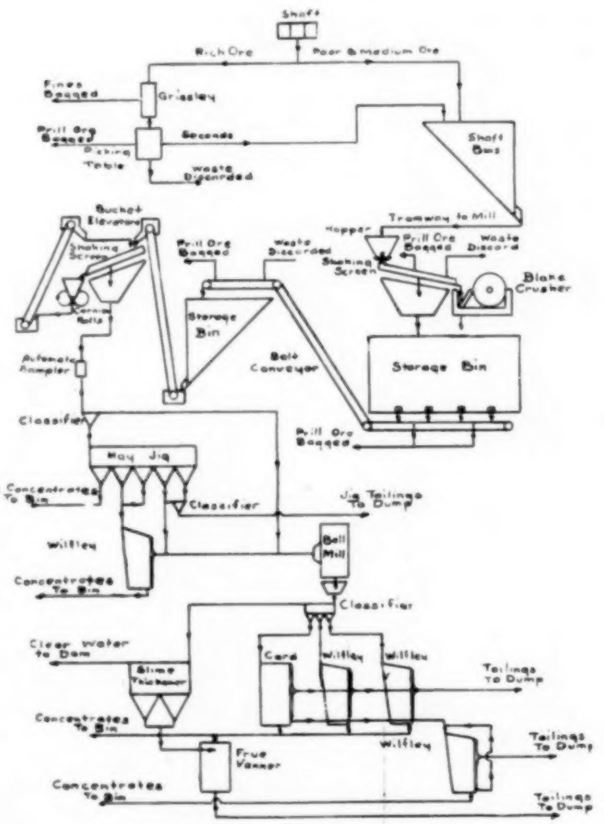


FIG. 1.—ORIGINAL CONCENTRATION PRACTICE.

This class of ore gave considerable trouble at the mill, but, fortunately, was very limited in quantity. A sample of rich ore assayed copper, 21.2 per cent; iron, 24.3 per cent; silica, 28.9 per cent; sulphur, 25.1 per cent.

Original Concentration Practice

The concentration in the original mill was effected by hand-picking, jigging, and tabling. A considerable portion of the copper mineral arrives at the surface in the form of large lumps of very pure quality, eminently suitable for sorting out by hand.

The rich ore is tipped over a grizzly at the shaft bins; the over-sized hand-picked, and the undersize bagged direct.

The medium and poor grades are tipped directly into the bin and trucked thence to the mill. The trucks discharge into a hopper, from which the ore is hand-fed through a chute to a shaking screen (1⅜-in. round holes) on which it is hand-sorted. The oversize goes to a 15-in. x 9-in. jaw crusher of the Blake type, and is reduced to 1-in. ring size. The undersize from the screen and the crushed ore fall into a bin, and is fed by push feeders to an 18-in. belt conveyor, on which it is hand-sorted by boys, both prill ore and waste being picked out. The conveyor discharges into the mill storage bin, of capacity sufficient to keep the mill going for 24 hours.

The crusher, belt conveyor, and pickers only work day-shift.

From the mill storage bin the ore is fed by a push feeder to bucket elevator, discharging on to a shaking screen (⅛-in. holes, standard spacing). The oversize goes to 30-in. Cornish rolls, and the product from the rolls is elevated by a bucket elevator back to the screen. The undersize from the screen goes to a classifier; spigot to four-compartment May jig.

When the tables were in use the product from No. 1 jig hutch ran to concentrate bin; Nos. 2 and 3 hutch spigots to a No. 5 Wilfley table; concentrate to bin; tailing to No. 5 Krupp ball mill; No. 4 hutch spigot direct to ball mill; jig tailing to classifier; spigot to waste as jig tailing, overflow to ball mill.

The ball mill crushed through steel screens with eleven 3/64-in. round holes to the linear inch.

From ball mill the pulp ran to a three-compartment classifier: first compartment spigot to No. 5 Wilfley; second to another No. 5 Wilfley; third to a Card table; overflow to a baffle board slime thickener; thickened slime to a Frue vanner; concentrate to bin; tailing to waste. The concentrates from the two Wilfleys and the Card table ran to bin; the middlings to another No. 5 Wilfley, and the tailings to waste. The concentrate from the last Wilfley ran to bin; the middling was returned over the same table, and the tailing ran to waste. Reference to the flow sheet (Fig. 1) will make the scheme clear.

With this arrangement the average extraction was about 74 per cent of the total copper in the feed to the mill. The tailings varied from 1.5 per cent to 1.8 per cent copper, the loss chiefly occurring in the fine and slime. The concentrates varied from 20 per cent to 21 per cent copper.

First Trial of Flotation

Considering the type of plant, these results were very satisfactory, but the advantages of concentration by flotation having been brought under the notice of the management, tests were carried out on samples of the milling ore by the Minerals Separation, Ltd., in Melbourne. The results were such that it was decided to replace the tables with a Minerals Separation unit.

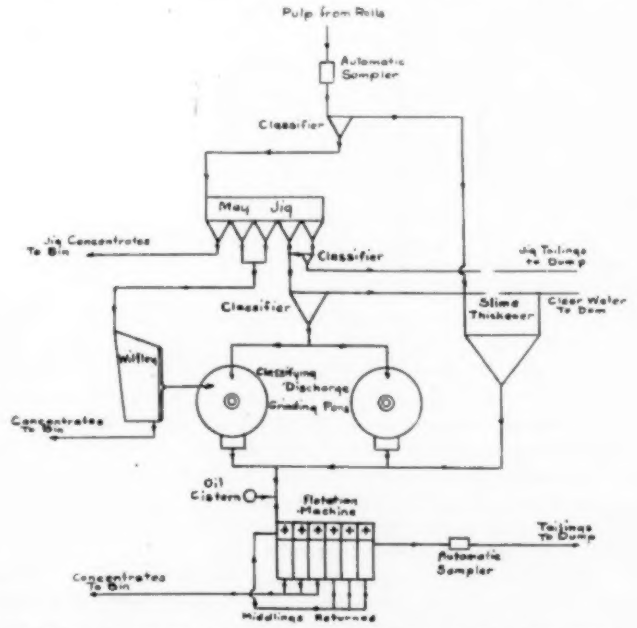


FIG. 2.—CHANGED CONCENTRATION PRACTICE WITH FLOTATION

The mill building was extended to house the new plant, which, in the original design, consisted of a baffle board slime thickener, two grinding pans, and a flotation machine. The departure from the old flow sheet commenced at the jig.

No. 1 spigot ran to concentrate bin as before; Nos. 2 and 3 to Wilfley table, and table concentrates to bin as before, but tailings to grinding pans. No. 4 spigot product of jig, and

*Excerpt from the *Proceedings*, Australasian Institute of Mining Engineers, No. 7, September, 1912.

classifier spigot product at foot of jig were sent to the pans, and thence direct to flotation machine. Overflow from classifiers at head and foot of jig was sent first to slime thickener and thence to flotation machine. The scheme is shown in flow sheet, Fig. 2.

The grinding pans were 8 ft. in diameter, running at 30 r.p.m., of the classifying discharge type.

Description of Flotation Machine

The flotation machine is of the standard Minerals Separation type, and is classed as a 12-in. unit. For the purpose of making

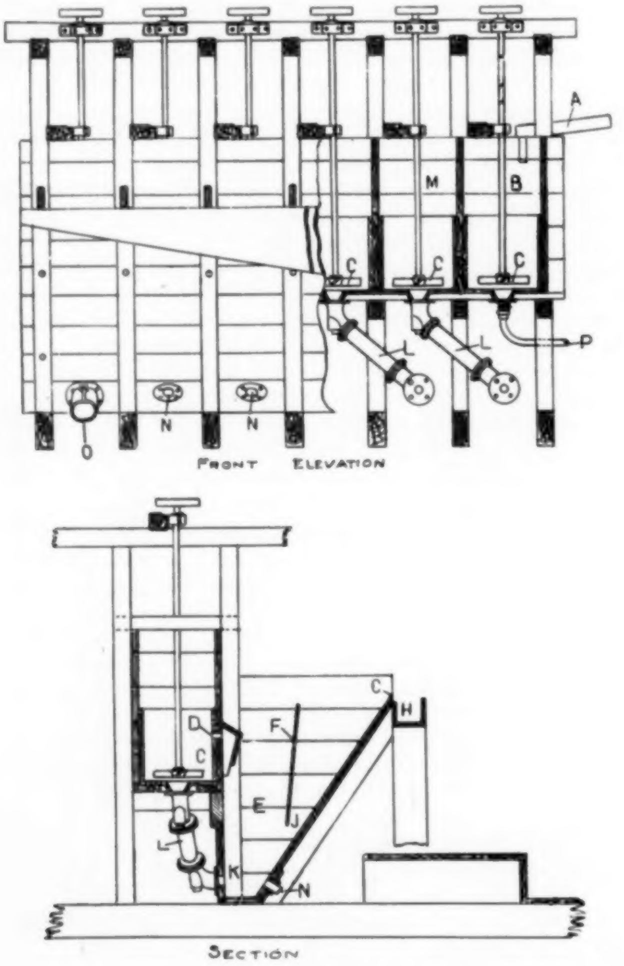


FIG. 3.—FLOTATION MACHINE

its operation clear to those unacquainted with it, the accompanying sketches (Fig. 3) are submitted.

The pulp with requisite amount of oil added, runs to machine by launder *A*, and discharges into No. 1 agitating box *B*, where it is violently agitated by stirrer *C*, revolving at about 450 r.p.m., with an upward throw.

The aerated pulp escapes through a slot *D* (section), and is deflected downwards into hutch *E*. Bubbles with sulphides attached rise against baffle *F* and form a dense froth on the surface of the hutch, which gradually moves forward and overflows at lip *G* into launder *H*. Any gangue particles attached fall out and pass downwards through space *J* to mix with the remaining pulp and pass out through orifice *K* into pipe *L*, leading to the bottom of No. 2 agitating box *M*.

The overflow from lip *G* is controlled by regulating the size of orifice *K* by means of a suitable valve. At the bottom of each hutch is a hole *N* blocked by a wooden plug or suitable valve, for sluicing out the contents of the hutch when closing down. At the elbows of pipes *L*, are similar holes and plugs for the same purpose.

In No. 2 agitating box *M* the operation is repeated, and so on throughout the six boxes and hutches. The residue is discharged from the bottom of No. 6 hutch through valve *O*, which is controlled by suitable means so that the overflow from the lips of the six hutches can be adjusted to what is required.

The stirrers *C* are either driven from a countershaft by quarterturn belts or else by bevel gears, the latter being positive in their action, but objectionable on account of noise and excessive wear and tear.

If required, a middling can be made and returned to the bottom of No. 1 agitating box through pipe *P*.

First Results Unsatisfactory, Necessitating Alterations

When put into operation the results from this plant were not quite satisfactory, and two members of the Minerals Separation staff were sent to investigate. Many minor faults were disclosed, which did not permit proper regulation and adjustment of the machine. The plant was closed, the machine overhauled and radically altered.

The slicing valves, operated by a lever to close or open the mouths of the pipes leading from the bottom of the hutches to the agitating boxes, were replaced by simple flap valves operated through a ½-in. rod by a thread and hand wheel.

The slicing valve for regulating the discharge of tailings from the machine, originally operated direct by hand, was also altered so as to be controlled by hand wheel and thread.

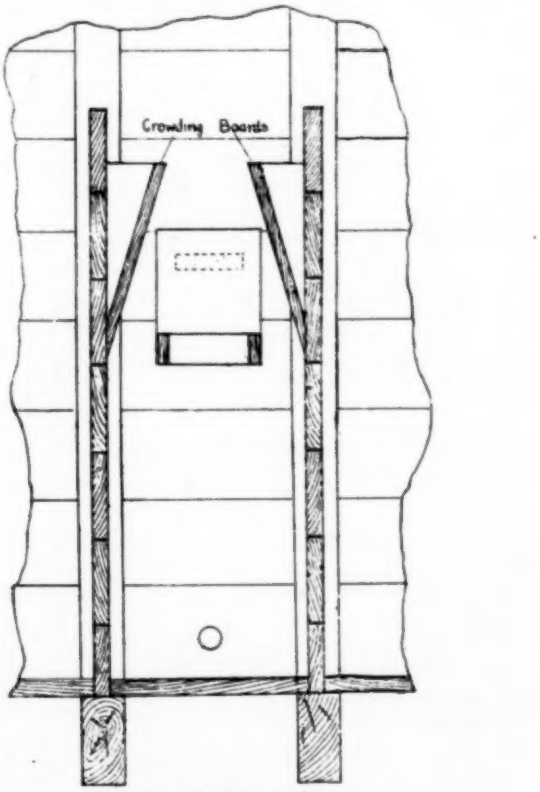


FIG. 4.—CROWDING BOARDS

This arrangement permitted very delicate adjustment of the overflow from the hutches, which was impossible under the old conditions.

The width of all six hutches being the same, the froth became very thin and poor on the last four. For the purpose of contracting the surface and giving greater thickness and body to the froth, "crowding boards" were put in, which gradually decreased the width of the hutch surface from 11½ in. on No. 1 to 4½ in. on No. 5. See Fig. 4.

With the idea of giving the sulphides less chance of dropping out of the froth, No. 6 hutch was shortened by replacing the baffle with a watertight bulkhead and leading the froth to the front of the machine in a launder. By this means the chance of mineral being lost in the tailings was considerably lessened as the settling space was halved.

The surface widths of the hutches were now as follows:

No. 1 .... 11½ in.
No. 2 .... 11 in.
No. 3 .... 7½ in.
No. 4 .... 5½ in.
No. 5 .... 4½ in.
No. 6 .... 4 in.

A sample of the overflow from each hutch gave the following results:

No. 1		No. 2		No. 3		No. 4		No. 5		No. 6	
Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Inso %
27.2	18.9	24.1	27.8	22.3	28.0	15.6	39.8	14.3	45.0	9.4	61.4

This test showed that the grade of concentrates from the last three hutches was not high enough to permit of them being sent to the concentrate bin, so arrangements were made for returning a middling to the machine.

The overflow from the last three hutches was fammed back by a bulkhead at the discharge end of the concentrates launder in front of the machine, and returned to No. 1 agitating box by a 1-in. pipe.

The concentrates from the first three hutches were carried by a tin tray over the bulkhead into another launder running to the concentrate bin.

To get at the insides of the agitating boxes the machine practically had to be dismantled. To obviate this, a rectangular hole was cut in the back of each box, so as to leave about 1½ in. of timber on each side, for the purpose of bolting on a door. The insides of the boxes were now readily accessible, and alterations could be carried out.

Examination showed that the 1½ in. redwood had worn badly, although the machine had only been in operation a few weeks. Liners were made from ¼-in. steel plate and bolted to sides and bottom in order to protect the woodwork from wear.

The orifice in the bottom of each box was at the same time made bell-mouthed, as it was found that by this means the suction caused by the stirrer could be largely increased.

After these alterations had been completed the plant was restarted, with the following results:

Run	Feed % Cu.	Concts. % Cu.	Tails. % Cu.	Extraction %
1	5.1	20.1	2.0	67.5
2	4.4	24.2	2.3	52.7
3	4.0	19.1	1.25	73.8
4	4.7	19.0	2.2	60.1
5	4.0	23.6	1.5	66.7
6	4.1	23.5	2.25	49.8

Formula for Calculating Results

The figures given refer to the machine considered as an isolated unit, and the extractions are the theoretical extractions worked out from the assay values of the feed, concentrate and tailing according to a method which was as follows:

Let x = weight in tons of concentrate when 100 tons of feed is treated.

" a = percentage of copper in feed by assay.

" b = percentage of copper in concentrate by assay.

" c = percentage of copper in tailing by assay.

Then

$(100 - x)$ tons = weight of tailing produced.

Now wt. of Cu. in feed = weight of Cu. in conct. + wt. of Cu. in tailing.

Therefore a ton of Cu. $= \frac{bx}{100}$ tons Cu. $+ \frac{c\,(100-x)}{100}$ tons Cu.

Putting this as an algebraic equation and solving, we get:

$$\frac{bx}{100} + \frac{c\,(100-x)}{100} = a$$

$$x = 100\,(a-c)/(b-c) \text{ tons.}$$

Hence wt. of Cu. in concentrate $= b\,(a-c)/(b-c)$ tons.

Hence percentage recovery $= 100\,b\,(a-c)/[a\,(b-c)]$.

By substituting the assay values for the symbols in this formula, the percentage recovery can be worked out very readily without knowing any actual weights. It was found exceedingly useful throughout the investigations into the working of the plant, and was used not only for estimating extractions, but also in the solving of other problems, such as the most economical grade of concentrate to produce.

A grading analysis was carried out to discover where the losses occurred. The results were as follows:

Mesh	Feed		Concentrate		Tailing		Extraction %
	Weight %	Copper %	Weight %	Copper %	Weight %	Copper %	
+ 20	1.52	2.2	0.18	17.5	1.85	2.0	10.3
+ 40	13.27	1.2	0.3	17.0	17.53	1.2	4.4
+ 60	15.20	0.8	0.31	18.5	20.77	0.7	11.7
+ 80	3.55	1.15	0.52	24.7	6.10	0.6	48.9
+100	22.5	1.4	7.42	23.6	21.4	0.6	58.6
+130	3.62	3.8	1.55	21.2	4.2	0.5	71.4
−130	40.4	5.45	89.7	23.75	28.0	1.2	77.9

Close Sizing Required

These figures showed the necessity for finer grinding and closer positive sizing of the feed to the machine. The presence of coarse particles is even more injurious to the result than the figures show, as the coarse sulphides, in falling through the froth, detach some of the finer material which would otherwise float off. For the purpose of finding out what work the grinding pans were doing, a grading analysis was made, with the following results:

Mesh	No. 1 Pan Feed		No. 1 Pan Discharge		No. 2 Pan Feed		No. 2 Pan Discharge	
	% Wt.	% Cu.	% Wt.	% Cu.	% Wt.	% Cu.	% Wt.	% Cu.
+ 16	16.4	2.3	.....	.....	10.0	1.4	.....	.....
+ 30	33.9	1.6	10.0	1.7	19.7	1.0	0.3	.....
+ 40	13.0	1.3	17.1	1.2	10.5	1.2	11.0	1.4
+ 60	18.3	1.2	31.9	1.4	18.0	1.5	30.9	2.0
+130	6.9	2.1	16.9	3.2	17.0	3.0	23.4	4.1
−130	11.5	3.7	24.1	5.0	24.8	6.7	34.4	6.2

Depth of discharge:—No. 1 pan, 21 in. above top of dies.
No. 2 pan, 22 in. above top of dies.

With the idea of reducing the +40 product, the lips of the pans were slightly raised and a sizing test made, which showed that the quanity of slime (—130) produced was largely increased, while the +40 product was only slightly reduced.

In spite of improvements in the flotation machine, difficulty was experienced in regulating the overflow from the hutches, and consequently the quality of the work done depended largely on the skill and agility of the man in charge.

The trouble appeared to be due to two causes:

(1) Fluctuation in the quantity of feed coming to the machine.

(2) Fluctuation in the speed of the stirrers.

The latter were driven from the main mill engine, the speed of which altered slightly according to the load on the pans and other units.

The froth on the hutches continued to be thin, with little body and holding power. This was probably due to three causes:

(1) The aforementioned fluctuations in the overflow.

(2) Excessive amount of oversize in the feed.

(3) Excessive dilution of pulp in the machine.

A test showed the latter to be 10 to 1, and owing to the arrangement of the mill no water could be cut out of the circuit.

A theory subsequently exploded was advanced that the poor froth was due to insufficient sulphide mineral in the pulp, and

the return of the overflow from the fourth, fifth and sixth hutches was expected to rectify the trouble by keeping a certain amount of mineral in circulation.

Final Rearrangement of Mill

It was finally decided that satisfactory work was impossible under the existing conditions, and the plant would have to be closed down and rearranged. The new plant was to embody the following improvements:

(1) Alteration of pans from classifying discharge to positive central feed type.

(2) Positive screening of all feed to machine.

(3) Thickening feed to machine, at the same time forming a reservoir from which a constant supply could be drawn.

(4) Independent motive power for stirrers, so that speed could be kept constant.

(5) Apparatus for regulating and keeping constant the supply of oil to the pulp.

The flotation plant was closed and the tables re-started. Plans and designs were prepared along the above lines, and the alterations carried out, occupying about eight weeks.

In the rearranged plant tabling is done away with altogether. No. 1 jig hutch spigot runs to concentrate bin, as before. Nos. 2, 3 and 4 spigots to classifier; spigots to pans; overflow to thickening box before machine; jig tailing to classifier; spigot to waste (temporarily); overflow to pans. The feed is equally distributed between the two pans by a distributing box. Discharge goes from pans to a bucket elevator, thence to revolving screens; undersize from screens to thickening box before machine; oversize back to pans. The overflow from classifier at head of jig also ran on to the screens.

The spigot of the thickening box ran to machine; overflow to slime thickener; thick slime to machine; clear water to dam. The oil dropped from a cistern into the launder leading to the machine. Reference to the flow sheet (Fig. 5) will make the scheme clear.

The only entirely new units were the revolving screens and the bucket elevator; the rest was merely a re-arrangement of existing plant.

The elevator was constructed with wooden boot of ample size to prevent excessive wear; 3-ft. pulleys, 22-ft. centers; 12 in. x 8-ply Balata belt, traveling 300 ft. per minute, with 10 in. x 6 in. x 6¼ in. buckets spaced 15 in. apart; lip of discharge 6 ft. below center of top pulley, to allow buckets to clear themselves, and each bucket had three 3/16-in. holes bored in the bottom to assist in this. After the buckets wore smooth this elevator did very good work, and so far has given next to no trouble. The screens are revolving trommels of the improved King type, driven by worm gearing so as to revolve at 11 r.p.m. They turn in opposite directions, and are fed from a distributing box placed between them, having two discharge lips, designed to spread the pulp evenly over the screens. By slipping a board in, the pulp can be cut out from either screen at will. The worm gears are attached to the shaft by a clutch, so that either screen can be stopped when necessary. This arrangement is very compact and neat. The trommels take 36 sq. ft. of wire screen to cover them, and the screen cloth lasts from ten to twelve weeks, according to the mesh and gauge.

Screen Analysis

The screens used were Greening's L. W. B. (600 mesh, 0.0164 in. diam. of wire, 0.0244 in. aperture) on one trommel, and Greening's A. K. D. screen (900 mesh, 0.0116 in. diam. of wire, 0.00217 in. aperture) on the other.

A grading analysis of the product from the A. K. D. screen was as follows:

Mesh	% Weight	% Cu.	Mesh	% Weight	% Cu.
+40	0.8	1.2	+100	7.7	2.7
+60	18.2	1.4	+130	2.6	2.8
+80	22.9	2.1	−130	47.8	4.8

The product from the L. W. B. screen gave the following figures:

Mesh	% Weight	% Cu.	Mesh	% Weight	% Cu.
+40	0.7	0.8	+100	8.5	3.3
+60	29.0	1.2	+130	4.3	3.7
+80	25.7	2.3	−130	31.8	5.3

These figures were considered more satisfactory, so henceforth L. W. B. trommel alone was used.

The alterations to the pans consisted in the removal of the baffles and classifying discharge lips, enlarging the discharge holes, and placing a cylinder round the center of the pan with watertight joint at bottom, to take the central feed.

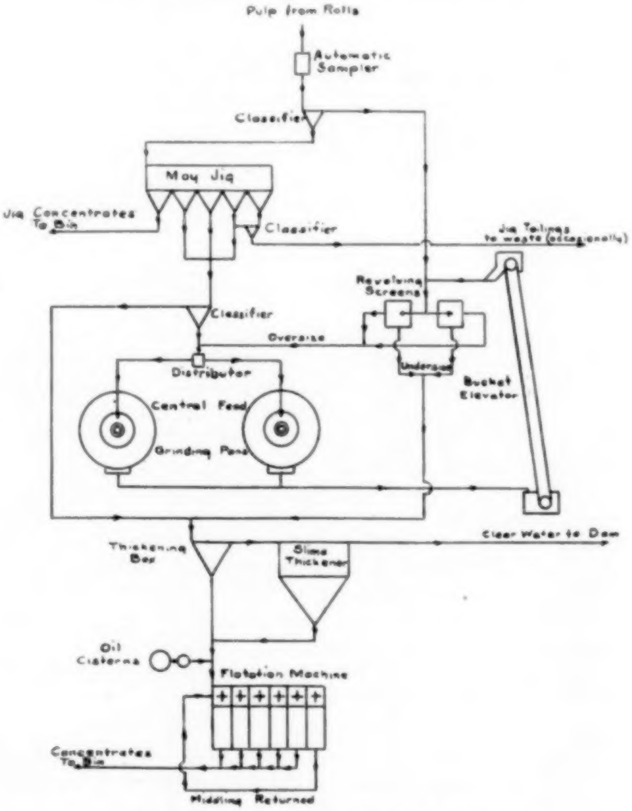


FIG. 5.—TRIAL ARRANGEMENT OF MILL.

It was noticed that, unless the pulp circulated freely from periphery to center by way of the curved channels between the shoes and dies, the pans did bad work, and the launders and screens soon became overloaded. The discharge was baffled to 7 in. above the tops of the dies in the hope that the head thus created would assist the circulation. This resulted in the pans choking so hopelessly that the engine stopped. The pans were then thoroughly cleaned out, the baffle reduced to 4 in., the shoes put hard down, and more water added to the feed.

This brought about the desired result, and the pans did splendid work, as the following grading analyses show:

Mesh	Pan Feed		Pan Discharge	
	% Weight	% Copper	% Weight	% Copper
+ 16	27.8	1.1	0.4	1.1
+ 20	10.2	1.2	1.8	1.0
+ 40	23.2	1.2	12.0	1.0
+ 60	21.2	1.3	34.0	1.1
+ 80	8.5	2.5	25.3	1.6
+100	1.8	4.3	7.4	2.7
+130	0.9	4.8	2.9	3.1
−130	6.4	5.1	16.2	3.8

When the sample was taken all the jig tailings were running to the pans. The mill has a capacity of about 60 tons per 24 hours, so, under these conditions, after allowing for jig concentrates and slime eliminated in classifier, each pan was treating pulp at the rate of 25 tons per 24 hours and consuming about 10 hp. in the operation.

Using these figures in conjunction with the above grading analysis, the relative grinding efficiency, according to Stadler's method, works out at 10.4.

Unfortunately, with the pans running under the above conditions (*i.e.*, shoes hard down) the load on the mill engine was too great, and it had a tendency to slow up, so the shoes had to be eased a little. This had the effect of interfering with the circulation in the pans, and a grading analysis showed the relative efficiency to have dropped to 2.6.

Owing to the lack of power the jig tailings had to be run to waste for about 3 hours out of the 24. This trouble has now been overcome by the installation of a small auxiliary engine.

Consistency of Pulp for Flotation

A pulp going about three of water to one of solids can be run to the flotation machine under the following conditions:

Thickening Box.	Slime Thickness.
Depth of discharge, 2 ft.	5 ft. 8 in.
Size of spigot, ⅝ in.	¾ in.

If necessary, it can be brought as low as two of water to one of solids by altering the spigots and depths of discharge. The dilution in the flotation machine itself is always higher than that of the pulp fed to it, on account of the large proportion of the water in the middling returned. The oil-feeding apparatus consists of two cisterns, one at the back of and above the other. The top one is fitted with a float valve, the float of which rests on the surface of the oil in the one below, and by this means the depth of oil, and likewise the pressure at the tap is kept constant. The drops from the tap are found to remain remarkably constant, except when dirt or water is allowed to get into the oil.

The engine driving the stirrers is a small 6-in. x 6-in. vertical Tangye engine, running at 250 r.p.m. and 100 lb. boiler pressure. It runs light, developing about 4 hp., and is governed by an ordinary centrifugal ball governor of Pickering type.

The cast-iron stirrers last from 4 to 5 months, and are screwed to the end of the spindles, the direction of rotation tightening them.

The only alterations to the flotation machine were in connection with the returning of the middling and the removal of the concentrate. The middling return pipe was increased to 2 in. in diameter, and holes were left at the bends to enable the pipe to be cleaned out when choked, by inserting a length of iron wire. This operation was very seldom necessary.

Another launder was added to the front of the machine, the inside one carrying the middling and the outside one the concentrate. Removable tin trays carry the concentrate across to the outside launder, and when necessary the tray is removed and the overflow from any hutch is allowed to return with the middling. With the old arrangement enough fall could not be given to the tray carrying the concentrate, and in consequence it had continually to be cleared with a scraper.

When put into operation the new plant soon justified the alterations. Once regulated, the overflow from the hutches remained steady and constant for an indefinite period, provided nothing went wrong with the rest of the mill.

On several occasions the flotation machine ran for a couple of days without any attention other than oiling. When all the jig tailings were being treated the concentrate from No. 4 hutch was so rich that it was run direct to the concentrate bin, and by adjusting the machine the grade of No. 5 could be raised sufficiently to do the same with it. This left the overflow from No. 6 hutch alone to be returned as a middling.

Strange to say this alteration had no apparent effect on the grade of the bulk concentrate produced, and certainly no deleterious effect on the efficiency of the machine.

The re-agent used is *Eucalyptus dives* oil, distilled in the bush country at no great distance from the mine. It is fed to the launder leading to the machine at the rate of 12 cubic centimeters per minute, which works out at a little under ½-lb. per ton ore. It costs 8d. ($0.16) per lb. delivered at the mine. No acid at all is used, one attempt to do so proving disastrous.

Effect of Oxidized Ore

A certain class of ore frequently gave trouble at the machine, and when present in the feed invariably decreased the extraction. This type of ore was that mentioned at the beginning of these notes—*i.e.*, decomposed, oxidized, and kaolinized lode matter.

On one occasion, when the pulp was discolored from a whitish-grey to a dirty yellow by this material, the process became inoperative, no true froth forming whatsoever, and the pulp leaving the machine at practically the same value at which it came in. Large slimy bubbles and small masses of coagulated sulphides covered the hutches.

The bubble films were very tenacious, the bubbles clinging to the sides of the hutches and allowing the turbid water to flow away from beneath them.

No improvement took place while the pulp showed any signs of discoloration. Salts in solution, or acidity of the pulp and excessive fine slime were considered likely causes of the trouble. The above extreme conditions were never again encountered, but on several occasions trouble was experienced with an excessive quantity of froth, made up of large bubbles with slimy, cohesive films.

A sample of pulp taken while these conditions ruled gave the following results:

(1) The decanted solution contained 36 grains to the gallon of salts in solution.

(2) The decanted solution was neutral to litmus.

(3) 52 per cent of the solids in the pulp passed 130-mesh sieve.

(4) The —130 product assayed 0.11 per cent of oxidized iron.
The +130 product assayed 0.03 per cent of oxidized iron.
The bulk sample assayed 0.07 per cent of oxidized iron.

(NOTE.—The oxidized iron was taken as a measure of the oxidized material.)

After this each daily sample of feed was assayed for oxidized iron, but no definite relation could be detected between the amount of oxidized iron present and the result of the day's run; but amounts above 0.05 per cent seemed to have bad effects, and when above 0.07 per cent the results were invariably poor.

The deleterious effect of the oxidized and decomposed material seemed to be to its physical rather than its chemical properties, and the peculiar nature and excessive quantity of the slime formed from it appeared to be responsible.

Favorable Percentage of Slime

Daily grading analyses of the feed, extending over a lengthy period, showed that more than 40 per cent of slime was undesirable, the best results being obtained with from 25 per cent to 35 per cent.

Sometimes the presence of even undecomposed felspar in the gangue appeared to be responsible for a dirty, slimy froth.

At Kyloe the ideal ore seems to be a clean sulphide mineral in a clean quartz gangue. Given a feed approaching this ideal, the flotation machine is capable of marvelous work, as the following figures will show:

Feed % Cu.	Concentrate % Cu.	Tailing % Cu.	Extraction %	Ratio of Concentration
3.8	25.1	0.95	77.9	6.6 to 1
3.0	23.3	0.85	74.3	7.7 to 1
3.0	24.2	0.65	81.0	8.2 to 1
3.15	23.7	0.55	84.5	7.5 to 1
4.1	25.6	1.05	77.5	6.2 to 1
4.4	24.6	0.9	82.5	5.6 to 1

These figures represent consecutive daily runs on good clean

ore, the tailing sample being taken automatically. The machine is considered as an isolated unit. It can be safely asserted that no other concentrating apparatus yet devised can show better or even equally good results on feed which contains particles from 40-mesh up to the very finest slime.

After the flotation machine had been running about 5 months the boxes began to leak, and examination showed that the wood work had been worn away behind the liners. The bottoms and sides had to be renewed, and to save similar trouble in future the ¼-in. steel liners were replaced by ½-in. cast-iron ones, designed with interlocking joints so as to protect the woodwork entirely from wear. These liners can easily be removed and renewed, and have given no trouble.

The concentrates from the machine are run to bins, where they are settled, and the persistent froth beaten down with a water spray. Certain classes of slimy froth give trouble by building up and overflowing the bins. When the concentrates settle the flow is diverted to another bin, and the water syphoned off.

The sloppy mass is then shovelled into barrows and wheeled to the drying floors, which consist of thick cast-iron plates under which hot gases from a furnace flow. Owing to the high percentages of water the concentrates take some considerable time to dry.

Filtering was attempted, but, owing to the slimy nature of the concentrates, was not a success, and the output was not large enough to warrant the installation of some form of pressure filter.

When dry the concentrates are shovelled from the floors into bins, and bagged for despatch to the smelter.

The following figures show the recovery made by the mill each month, calculated from the assay value and weight of feed and the value and weight of concentrate despatched:

Month	Average Value of Tailing % Cu.	Extraction %
1st	1.26	82.7
2nd	1.11	86.1
3rd	1.03	85.4
4th	1.0	87.0
5th	0.95	93.0
6th	0.89	90.5
7th	0.81	85.0
8th	0.95	95.0
9th	....	....

The accepted assay value of each lot of concentrate sent to the smelter is given below:

Lot No.	Accepted Assay Value % Cu.	Lot No.	Accepted Assay Value % Cu.
1	23.2	7	25.27
2	26.17	8	25.24
3	25.95	9	25.92
4	25.05	10	26.00
5	25.52	11	25.06
6	25.66	12	25.48

General Results Improved by Flotation

These figures show an increase of over 10 per cent in the recovery and a little under 5 per cent in the grade of the concentrate, over the old system of tabling. The importance of the latter may be gauged from the fact that after deducting freight and smelting charges, with copper at £50 ($250) a ton, a ton of copper in a 20 per cent concentrate is only worth £28.76 ($144) to the company while a ton of copper in a 25 per cent concentrate is worth £32.52 ($162)— an increase of £3.76 ($18) per ton of copper, representing on a 1000-ton output £3,760 ($18,800) a year. The average cost per ton milled for the twelve months previous to the installation of the flotation plant was 7.81 shillings ($1.90). The average cost since the installation is 8.21 shillings ($2.00)—an increase of 4.8 pence ($0.10) per ton milled, or about £370 ($1,850) on the yearly output.

The grade and tonnage of the ore treated was about the same for 1911 when tables were used, and 1912 when flotation was adopted, thus affording an excellent opportunity for comparing results under the old and new systems of milling.

Copper is taken £60 ($300) a ton in both cases.

The figures show that, while the expenditure increased £1,300 ($6,500) the revenue went up over £4,300 ($21,500), leaving over £3,000 ($15,000) to the credit of the flotation plant.

The cost of construction and alterations came to about £1,300 ($6,500) so that the new machinery has paid for itself more than twice over during the first year of operation.

In addition to this, it is only fair to the flotation plant to draw attention to the fact that disbursements and maintenance on the old units have been larger than they were during 1911, owing to depreciation with age.

Nichols-Hesse Dinner

By the joint action of the executive officials of the American Chemical Society, American Electrochemical Society, American Institute of Chemical Engineers, Chemists' Club, the New York Section of the Society of Chemical Industry and the New York Section of the Verein Deutscher Chemiker, it is proposed to give a complimentary dinner to Dr. W. H. Nichols and Dr. B. C. Hesse as a mark of appreciation of their work in connection with the Eighth International Congress just closed. Professor Edward W. Morley, as the honorary president of the Congress, has been asked to preside at this dinner.

It is proposed to hold the dinner on April 19 at the Chemists' Club, the price to be $3.00 per plate without wine. Rumford Hall will accomodate hardly much more than 250 guests, and the men who intend to go to this dinner should make early reservations.

Milwaukee Meeting of the American Chemical Society

The forty-seventh annual meeting of the American Chemical Society will be held in Milwaukee, Wis., from March 25 to 28. Headquarters will be at the Hotel Pfister. The meetings will be held at Marquette University. A feature of the convention will be an excursion to Madison, Wis.

Atlantic City Meeting of the American Electrochemical Society

The spring meeting of the American Electrochemical Society will be held in Atlantic City, N. J., from April 3 to 5. (Thursday to Saturday of the first week in April). Headquarters will be at the Hotel Traymore.

A meeting of the board of directors will be held on the evening of Wednesday, April 2. The convention will start on Thursday, April 3, with the annual business meeting which will be followed at once by a meeting devoted to the reading and discussion of papers, to be continued in an afternoon session. For the evening it is intended to arrange a smoker.

Practically the whole day of Friday (April 4) will be devoted to an excursion to various chemical plants in and near Philadelphia where the members and guests will be taken by a special train. After their return in the evening to Atlantic City, dinner will be taken and there will be a lecture with demonstrations by Prof. E. B. Kenrick of the University of Toronto.

The Saturday morning meeting will be devoted to the presidential address by Dr. W. Lash Miller which will be in form of an introduction to the symposium of papers on electroplating and electrodeposition of metals, while another professional session in the afternoon, devoted to a continuation of this symposium and to the reading and discussion of other papers, will conclude the meeting.

The fall meeting will be held in Denver, Colo.

Dr. W. Lash Miller, of the University of Toronto, is the president, and Dr. J. W. Richards, of Lehigh University, South Bethlehem, Pa., is the secretary of the American Electrochemical Society.

Symposium of Papers on Alumina

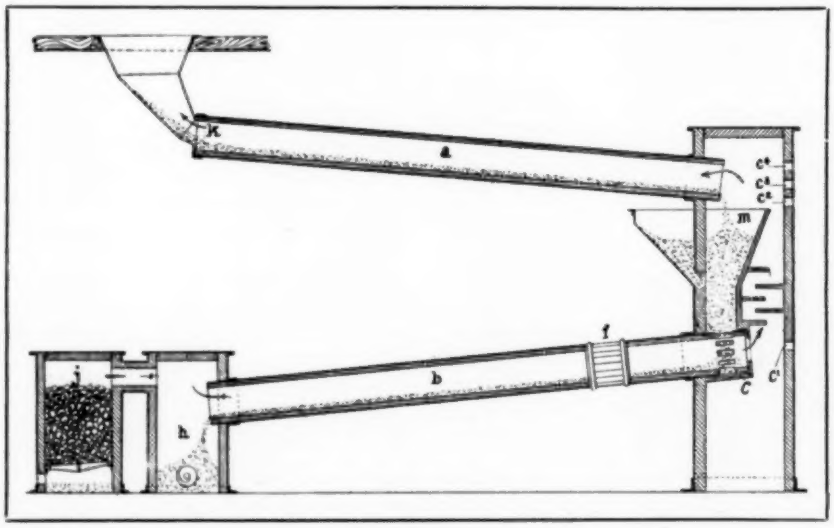
Four papers on the Production and Uses of Alumina, by J. W. Richards, S. A. Tucker, A. H. Cowles, and L. E. Saunders

A symposium of papers on alumina—its production and its uses—was the feature of a joint meeting of the New York Sections of the American Chemical Society, the American Electrochemical Society, and the Society of Chemical Industry, held on February 7, 1913, at the Chemists' Club in New York City.

The meeting was held under the auspices of the New York Section of the American Electrochemical Society and was called to order by the chairman of the section, Mr. **Elmer A. Sperry.** In a brief business meeting which followed, this section elected officers for the coming year as follows:

Mr. **Lawrence Addicks,** superintendent of the Chrome, N. J., works of the U. S. Metals Refining Company, was elected chairman, and Mr. **H. B. Coho,** of the United Lead Company, New York City, was re-elected secretary-treasurer.

Mr. E. A. Sperry then made some introductory remarks on the subject of alumina. He referred to the presence at the meeting of Mr. Bradley and Mr. Cowles, two prominent pioneers of the metallurgy of aluminium and sketched the effect which cheaper alumina would have on the industry of aluminium and aluminium alloys. He referred especially to the remarkable properties of duralumin for many purposes in the industries. Samples were exhibited.



Apparatus for carrying out Serpek process (*k*, supply of ground bauxite and exit of gases; *a*, kiln for calcining; *m*, hopper for supply of calcined bauxite to kiln *b*; *b*, kiln for nitride reaction with electric resistance furnace *f*; *h*, chamber for removal of nitride; *j*, gas producer; *c* chamber for deposition of silica; c^1, c^2, c^3, c^4, air supply inlets for combustion of gases. From Serpek patent 996,032).

Then followed two papers by Dr. **Jos. W. Richards,** of Lehigh University, and by Prof. **Sam. A. Tucker,** of Columbia University, on the production of alumina and ammonia by the Serpek process which is attracting so much attention at present.

The Serpek Process

By J. W. Richards

The production of aluminium nitride (AlN) on a large scale is proposed in the process of Ottokar Serpek, followed by the decomposition of the same by water or caustic soda solution, thereby liberating the nitrogen as ammonia and producing alkaline aluminate solution from which pure alumina can be obtained. This rather daring chemical proposition has directed attention to aluminium nitride, the conditions of its formation and its properties.

The existence of aluminium nitride was suspected for some time before it was isolated. A rather indefinite number of years ago, probably about 1890, the attention of the writer was directed by his father to the fact that when metallic aluminium in a melted condition was skimmed, and the skimmings or dross laid to one side, that on pouring water upon them while still warm these skimmings gave off an odor of ammonia. The explanation which was then figured out for this phenomenon was that the hot aluminium in the skimmings oxidized to alumina, and that the rather high temperature thus produced locally caused the aluminium also to unite with the nitrogen of the air and form the nitride. In other words, that some of the aluminium united first with the oxygen and immediately thereafter some also with the nitrogen of the air. On sprinkling with water the nitride would react according to the following reaction:

$$2\,AlN + 3H^2O = Al^2O^3 + 2NH^3.$$

In the light of subsequent investigation and the formation of aluminium nitride in other ways, it appears that the explanation given in these early days was correct and that aluminium nitride is formed directly under such conditions from metallic aluminium and the nitrogen of the air.

LeVerrier, later, proved the presence of traces of nitrogen in commercial aluminium by dissolving it in caustic potash and passing the nitrogen evolved into Nessler solution, obtaining from it the ammonia reaction. He thus proved the presence of nitrogen in the metal and suspected it to occur as aluminium nitride dissolved in the excess of metal. In further experiments he treated aluminium with a current of nitrogen gas until it was saturated, and noted that this produced considerable diminution in the tensile strengh, elastic limit and elongation of the metal.

Prof. Mallet later obtained aluminium nitride corresponding closely to the formula AlN by heating metallic aluminium in the carbon crucible to a moderate temperature for several hours in contact with dry sodium carbonate. Some alumina is formed, some sodium vaporizes and some carbon is deposited; the reaction is probably as follows:

$$3Al + Na^2CO^3 + N = AlN + Al^2O^3 + C + 2Na.$$

Aluminium nitride containing 34 per cent of nitrogen and 66 per cent of aluminium was obtained as little yellow crystals and amorphous drops. When calcined in the air it was found to lose nitrogen and form alumina; it was decomposed in moist air, losing its transparency, becoming lighter yellow, evolving ammonia gas, and finally leaving the alumina residue.

Caustic potash solution and melted caustic potash both attacked it actively, disengaging ammonia and forming potassium aluminate.

A later experimenter described the compound as a grey amorphous powder, but otherwise with properties similar to those described.

Coming now to the work of Serpek we find his first proposition to be the production of aluminium nitride from aluminium carbide, which is indeed a possible reaction. In his U. S. patent 867,615 (filed June 19, 1906, issued October 8, 1907) he describes the passing of nitrogen or a gas containing nitrogen over aluminium carbide as an initial material heated to red heat. He describes the volume of the carbide increasing as it is converted into nitride, and that the absorption of nitrogen was increased by diluting the carbide with such materials as carbon or alumina or aluminium chloride; also that traces of hydrochloric acid or sulphur dioxide in the gas currents facilitated the conversion. The claims of this patent are essentially for the method of producing aluminium nitrides by heating aluminimum carbide in an atmosphere containing nitrogen to a red heat with or without the diluting substances admixed and with or without traces of acid gas in the nitrogen used.

Attention may be called to the term "aluminium nitrides" in this patent, corresponding to the fact that in the product thus obtained by Serpek he found that part of the nitrogen in the product was driven off by contact with air, and a further part by boiling with water, and a still further part by treatment with caustic alkali solutions. These facts pointed to the non-homogeneity of the product, which thus appeared to consist of a mixture of different nitrides instead of one distinctive chemical compound.

Continuing his work we find that Serpek later concluded that it might be possible to obtain the nitride directly from mixtures of alumina and carbon, which it will be recalled he used as diluent substances in the conversion of the carbide. He was apparently not able at that time to produce nitride directly from these two substances, but found that in the presence of a small amount of metal such as copper or iron, which is capable of forming an alloy with aluminium the formation of nitride from the mixture took place. In his U. S. patent 888,044 (filed May 24, 1907, issued May 19, 1908), he describes mixing alumina with carbon in the proportions necessary for formation of CO gas (Al^2O^3 3C), adding about 5 per cent of either copper or iron or a mixture of the two, and heating the mixture to a red heat in a current of nitrogen gas, at which temperature the partial formation of an alloy of aluminium produced by reduction, with the copper or iron would be initiated. The idea seems to have been to produce a mixture on the point of reaction from its constituents and then to super-add the influence of the current of nitrogen gas. Traces of HCl or SO^2 gases are said to facilitate this reaction also, and the product is described as almost pure aluminium nitride. The reaction is described as developing a high temperature as soon as it is initated. The claims are for producing aluminium nitrides by heating the mixture of alumina with carbon and a metal capable of forming an alloy with aluminium, to red heat in an atmosphere containing nitrogen, with or without the presence of small amounts of acid gases.

It is evident that in this method the conversion of alumina into aluminium nitride was aimed at, with the assistance of small amounts of metals as catalytic or assisting agents. Whatever may have been the results of the previously described process it finally led the investigator in the end, to the method described in Serpek U. S. patent 987,408 (filed December 15, 1909, issued March 21, 1911) in which aluminium nitride is produced directly from Al^2O^3 and carbon in the presence of nitrogen, at a much higher temperature than previously recommended. This patent appears to be the result of considerable investigation on the conditions of this direct production and specifies very exactly the temperatures at which the direct conversion takes place. The patentee speaks of the general supposition that extremely high temperatures would be required for such direct conversion, such as the highest temperatures of the electric furnace, but explains that by careful investigation he had found that if the temperature is closely watched some combination of nitrogen with aluminium can be observed even at 1100 deg. C.; that at 1500 deg. nitrogen is absorbed fairly rapidly, at 1700 deg. energetically, and that at 1800 to 1850 deg. the reaction may be almost designated as violent producing almost chemically pure nitride. Further, Serpek found that temperatures higher than this gave a smaller yield, and that somewhere above 2000 deg. the production practically ceased, the decomposition temperature of the nitride appearing to be about 2120 deg. C. This brilliant piece of experimental work really laid the foundation for the Serpek process as it now exists. The inventor further discovered that impure alumina, such as bauxite ore, was converted at somewhat lower temperatures than the pure material, evidently owing to the catalytic effect of the impurities present.

The claims of this patent are for producing aluminium nitride by heating aluminous compounds and carbon, or more specifically alumina and carbon, in an atmosphere containing nitrogen to temperatures not exceeding 2000 deg. C., and preferably about 1800 deg.

Reference may very properly be made at this point to the experimental investigation and confirmation of these results by Prof. S. A. Tucker and Henry L. Read given before the Eighth International Congress of Applied Chemistry, published in its Transactions and in volume 22 of the Transactions of the American Electrochemical Society. This investigation, carried on in the electrochemical laboratory of Columbia University under carefully regulated conditions showed traces of nitrogen absorbed at temperatures of 1000 to 1100 deg. and as much as 30 per cent of nitrogen in selected portions of the product made between 1800 and 1900 deg. The product made at higher temperatures contained less nitrogen and was either sintered or fused. Impure alumina, such as bauxite, was found to be converted more easily than pure alumina, and the definite statements of the last Serpek patent were confirmed and verified.

Serpek's U. S. patent 996,032 (filed June 21, 1910, issued June 20, 1911), contained a detailed description of the apparatus intended for the commercial manufacture of the nitride from bauxite or other aluminium ores. The plant consists in two superposed rotating cylindrical kilns, similar in construction to a cement clinkering kiln. The kilns are inclined in opposite directions, so that the material being preheated in the upper kiln and passing from right to left for instance, falls into the opening of the lower kiln for treatment at higher temperature where it passes from left to right. Bauxite alone is passed through the upper kiln, and there calcined; then mixed with the necessary carbon in its passage from the upper to the lower kiln and the mixture treated by nitrogen at a high temperature in the lower kiln, which is provided with a detachable electric resistance furnace about midway of this length and which is intended to subject the charge therein to the reacting temperature about 1800 to 1900 deg. C. The material is discharged from the lower end of this kiln into an air-tight receiver. A large gas producer of the ordinary type furnishes producer gas (approximately ⅓ CO, ⅔ nitrogen) to the lower end of the lower kiln. This gas entering at a temperature of about 400 deg. becomes highly heated as it passes through the kiln in a direction contrary to the descending charge, and at the electrically heated zone at a temperature of 1800 to 1900 reacts upon the mixture forming nitride. After leaving the high temperature zone the gas enriched in CO coming from the reaction preheats the descending charge, issues from the upper end of the kiln into a vertical closed chamber and passes through it to the opening of the upper kiln where it meets a blast of air which burns it for the purpose of heating the contents of the upper or calcining kiln.

The apparatus as thus described achieves a methodical heating of the reacting mixture, utilizes the CO gas made in the producer and also that evolved by the reaction in the furnace; aims at an intimate mixture of the charge by the rotation of the cylinders and intimate contact of the nitrogen with the charge. The inventor states that the silica impurity of the

charge is mostly volatilized (probably as reduced silicon) from the charge at the reacting temperature and is thus carried out in the gases; and that the apparatus may be used for producing other nitrides made by analogous reactions.

The claims cover an apparatus comprising the features of a revolving inclined upper cylinder discharging into an inclined lower cylinder revolving in the opposite directon, through a vertical chamber in which the gas from the lower cylinder may be burned combined with an electric detachable electric resistance heating furnace within the lower cylinder. One claim is for the single feature of a rotary calcining cylinder carrying a detachable electric resistance furnace.

Undoubtedly a large amount of excellent investigation has been put upon this process, and it certainly deserves commercial success. The method is being tested in France upon a large experimental scale.

The Relation of the Production of Alumina to the Fixation of Nitrogen

By Samuel A. Tucker

The connection of nitrogen fixation and the production of alumina is to be found in the Serpek process and probably in this process only. The Serpek process is primarily one for the fixation of nitrogen, but necessarily involves the production of alumina as a by-product and this by-product is all important to the success of the process commercially.

Briefly the Serpek process takes alumina and carbon in the proportions to form aluminium nitride in a strongly heated atmosphere of nitrogen, or

$$Al_2O_3 + 3C + N_2 = 2AlN + 3Co.$$

The range of temperature necessary for this reaction is from 1600 deg. to 2000 deg. C. according to Serpek. The product obtained contains about 30 per cent nitrogen in the form of aluminium nitride, and is thus considerably richer in fixed nitrogen than calcium cyanamide which rarely runs over 20 per cent.

This reaction appeared so important to me that I thought it was worth while to investigate it and in conjunction with Mr. H. L. Read, the process was tried on a small scale. The results[1] of this work have been read at the recent Congress of Applied Chemistry, and it will be unnecessary to repeat them here, excepting that we found that the reaction worked satisfactorily on a small scale, the product contained 30 per cent of nitrogen, and that bauxite was the best material with which to supply the aluminum. The temperature is important. Below 1600 deg. practically no nitrogen is fixed, while above 2000 deg. C. decomposition is likely to occur. I would like to know more as to the influence of temperature, and I had expected to conduct some experiments on this point so that I might give you this information now, but an accident to the electric furnace prevented it.

The commercial product must be nearly pure aluminium nitride, and at first sight seems to consist of two forms, the blue and the yellow. Professor Luquer has examined these specimens microscopically and while his examination has not yet been completed, he tells me that the internal structure is the same, both are strong double refracting and show uniaxial structure. With our present information, it would be difficult to figure the cost of production of this material, and even the power necessary for its manufacture is probably not generally known.

The question of the necessary time to convert the charge is important and in our experiments we found that it had a great influence on the nitrogen content of the product. It, therefore, seems necessary that the mixture should be heated to a high temperature for a considerable time.

This is different from the Frank process for cyanamide in that the temperature for the actual nitrogen fixation is relatively low. Granting that the process can be worked out on a satisfactory cost basis, we see that the production of alumina is an all important consideration. By proper treatment of the Serpek product there seems to be no difficulty in obtaining the nitrogen as ammonia and the aluminium as alumina, and the latter should be practically pure Al_2O_3.

It might be supposed that the alumina so produced could be used over again in the process, but aside from its value in connection with the aluminium industry, pure alumina is not advantageous as a raw material, as it requires a higher temperature than bauxite. The process is, therefore, necessarily associated with the production of aluminium. If we might suppose that the Serpek process were to be conducted on the same scale as the Frank process for cyanamide manufacture, it is evident that it would involve the production of a large amount of alumina. From figures recently published regarding the Frank process, we find that the capacity output of all the plants covering various parts of the world for 1912 is roughly 200,000 tons of cyanamide. If we figure this as containing 18 per cent. nitrogen, it involves the fixation of 36,000 tons of nitrogen. With the Serpek product at 30 per cent. nitrogen content, it would require the production of 120,000 tons to equal this, and after decomposing to obtain the ammonia as follows:

$$2AlN + 3H_2O = 2NH_3 + Al_2O_3$$

we obtain 149,200 tons of alumina. This quantity of alumina should give 79,104 tons of metallic aluminium.

In the year 1911 we find that the capacity of all the plants producing aluminium was 62,100 tons, but that the estimated production actually was about 45,000 tons. We thus have raw material for the production of 34,000 tons of alumina in excess of present requirements. Of course, the process will not be conducted on such a scale for some time to come, and by that time there might well be a market for the alumina.

There appears to be no danger from the lack of bauxite, as I find that in this country 155,618 tons were produced in 1911 and 196,000 tons in France for the year 1910.

As the Serpek process would necessarily come into competition with the other means of fixing nitrogen, and particularly with the cyanamide process, it might be well to consider for a moment a few points in connection with the latter.

Briefly the Frank-Caro process for making cyanamide is to subject finely ground calcium carbide to the action of pure nitrogen at a temperature of from 1000 to 1200 deg. The energy necessary at this absorption stage is probably very small, as heat is evolved by the reaction:

$$CaC_2 + N = CaCN^2 + C$$

and the expenditure of energy is chiefly in the manufacture of the calcium carbide. Various conditions have to be observed in this absorption part of the process. The nitrogen must be practically free of oxygen and the temperature must not rise to the dissociation point of the cyanamide, which is probably about 1370 deg. C. The product so obtained contains anywhere from 15 to 22 per cent of nitrogen. The nitrogen is obtained by passing air over copper to take up the oxygen, or by fractionally distilling liquid air. Both systems seem to be in use, but with the latter some treatment of the resulting nitrogen with copper would probably have to be employed as nitrogen obtained from liquid air contains upwards of 8 per cent oxygen.

The process seems fairly complicated, but possesses the advantage over the high-tension arc systems of fixation in that a better return of nitrogen is obtained for the power used. In the Frank process the chief expenditure for power is in the formation of calcium carbide. The average of several plants making carbide is 5.2 kilograms of 80 per cent carbide per kilowatt-day of 24 hours. This gives 4.16 kilograms of pure calcium carbide, which corresponds to 173 grams per kilowatt-hour. Now if this is converted to cyanamide having a 20 per cent nitrogen content, we get 43.2 grams N. per kilowatt-hour.

The Birkeland-Eyde process obtains about 57.1 grams nitric acid per kilowatt-hour and the nitrogen content is 12.7 grams. We thus have about three and one-half times the nitrogen fixed by the Frank process as cyanamide as we do for the Birkeland-Eyde process as nitric acid, and this is probably the reason for its being established on such a large scale.

[1]See this Journal, volume X, p. 745 (1912).

As to the relative value of the fixed nitrogen fertilizers, we have no very precise information. Taking the various products adapted to this purpose in the different processes, we get calcium nitrate by the Birkeland-Eyde, calcium cyanamide by the Frank, and lastly ammonia by the Serpek process. These must all be compared with Chili sodium nitrate and the general opinion seems to be that the latter is the most valuable.

Field experiments have been conducted with cyanamide and calcium nitrate, but precise information is lacking. It would seem, however, from what has so far been done that we should put them in the following order: Chili saltpeter, calcium nitrate, cyanamide, ammonia.

The action of cyanamide is quite different from the others; it supplies a large amount of calcium which may or may not be an advantage. Its slow solubility is, however, advantageous.

Ammonia in the form of sulphate is pretty well understood in its relation to fertilization and for that reason the Serpek product should have a ready market in this connection.

* * *

The two papers by Professor Richards and Professor Tucker were discussed together. Some of the points touched upon in the discussion were the possibility of an intermediate reaction preceding the formation of the aluminium nitride, and the way how the impurities, especially silica, are removed in the process. Attention was called to the fact that French bauxite is in the average low in silicon and high in iron and American bauxite high in silica and low in iron, and that this might have some effect on the operation of the process with American bauxite.

After the conclusion of this discussion, in which some ten speakers participated, Mr. Alfred H. Cowles, of Sewaren, N. J., presented his paper on his own new alumina process, the fundamental points of which were already described by him in his paper before the International Congress of Applied Chemistry in September, 1912 (our Vol. X, page 659). Mr. Cowles' present paper contained, however, much that was new. We give it herewith practically in full:

Cheaper Alumina and Aluminium from Mineral Silicates

Through Reactions with Alkali Chlorides, Lime and Steam, and Between Florida Phosphate Rock, Potash Feldspars and Acids with the Production of Cheap Hydrochloric Acid, Potash Salts, Phosphate Fertilizers and the Production of Material for the Manufacture of Glass or Hydraulic Cement

By Alfred H. Cowles

Let two briquets be formed, one of kaolin or clay containing 19 parts of anhydrous pure clay and 23.4 parts of salt, and the other briquet containing the same relative proportion of salt and dry kaolin as the first, and fifteen parts of charcoal per one hundred parts of the mixture be present in the second briquet. If now these two briquets, each of the same size and form, be kept heated to a temperature equal to or just above the vaporizing temperature of salt, and if vapor of water in an oxidizing atmosphere surrounds them, it will be found that each briquet begins to be converted over its surface into a sodic-silico-aluminate which would have the formula of $(Na_2O)_2(SiO_2)_2Al_2O_3$ if the clay and salt be pure. The vibrating gaseous molecules of watery vapor and oxygen bring about this change over all exposed surfaces of the two briquets. This difference in the conversion will, however, be found. The briquet containing no charcoal will require nine times as much time for the conversion to penetrate to its center between walls, as the briquet containing charcoal.

This result is not self-apparent as the reaction requires an oxidizing atmosphere to oxidize the sodium produced before it volatilizes out of the briquet. That this result would occur, was based upon hypothetical reasoning on my part and its demonstration as a truth was brought about by experiments performed in the laboratory of The Electric Smelting & Aluminum Company at Lockport, New York. By coincidence Dr. Adolf Kayser was my assistant in performing the experiments.

Dr. Kayser in the early '90's, attempted to develop this reaction commercially using briquets containing no charcoal in large kilns of a down-draft nature. The work had proven a failure as had also the work of William Gossage in 1862, and that of Gruneberg and Vorster about 1873-76. During these experiments Dr. Kayser was of the opinion that they could not succeed because, as he thought, it would produce reducing conditions. The experiments, however, proved that the conversion penetrated the briquets nine times faster with charcoal in them, than was the case when briquets contained no charcoal. This result taught that it was necessary to expose large surfaces or porous masses to bring about a rapid conversion. This could be effected in more than one way, but the problem of cooling the gases and simplifying the elimination of dust led to the adoption of a tunnel furnace of the Gröndal type, with modifications to render it suitable for the process.

The fact that the reaction could be made to proceed rapidly opened up an entirely new vista, one of assured success to the process. It meant that the column of fuel gases and nitrogen necessary to pass through the charged furnace would become so far reduced as to permit of efficient commercial condensation of the hydrochloric acid evolved. It meant the employment of cheap fuel, sawdust or charcoal made from sawdust with possible by-products from the same, and far greater capacity of output with the same cost of plant. It meant that the process could be applied to other aluminous materials, such as potash feldspar mixed with chloride of potash or salt, and securing the chemically equivalent reaction.

Dr. Kayser had evolved a method of opening or rendering leachable alkali-silico-aluminate containing sufficient alkali, which means nearly one molecule more than that which occurs in feldspar. This he did by heating to a sintering temperature two molecular weights of lime to each molecular weight of silica in said compound, after which, the product may be leached and the alkali aluminate very effectively removed therefrom. Our company acquired this patent from him.

The process is easily understood. In the manner in which we are attempting to perfect it at Sewaren, N. J., in our present work, briquets are formed by the same clay-working machinery that produces what is known as hollow-ware or conduit bricks. When the clay, salt, and sawdust are mixed together and water is added, this mixture acts almost the same as clay as to plasticity in passing through the briquetting machinery, but unlike the clay brick, it does not shrink or expand from the period of its first formation during drying and until it is formed into a finished product containing soda, silica, and alumina.

These briquets are loaded onto flat cars, the surface of each car as now used being 12 ft. long and 5½ ft. wide, holding fifty-five bricks. (See Fig. 3, p. 661 of Vol. X of this journal.) These loaded cars are passed first through a drying furnace where the free water is driven off, and then by a transfer track, into a vestibule connected with the long kiln. (See Fig. 4, p. 661, Vol. X.) By a hydraulic pusher, after the door of the furnace is opened, and the door of the vestibule is closed, the loaded car is gradually pushed into the charging end of the furnace, and a train of fourteen cars in front of it are also gradually pushed forward towards the discharging end of the furnace, while a finished carload is coming out at that end.

The cars are covered with firebrick, the latter of a nature to protect them from being materially attacked by salt vapors. A sand seal extends the whole length of the furnace, and pure air, by a blower, is blown under the train of cars in such a manner as to prevent any leakage of hydrochloric acid fumes downward. In operation, the temperature of all exposed iron parts of the furnace are kept above the boiling point of mixed acid and water, and as is well known, at this temperature vapor of hydrochloric acid does not attack iron.

Knowing the composition of the clay and the percentage of salt in the briquets steam is blown into the discharge end of

the furnace, and the rate by pounds weight of steam per minute is read by an indicating steam-flow meter. Suppose this rate is adjusted to the rate of feed of the material on the cars to effect the decomposition of salt with the formation of, say, 33 per cent hydrochloric acid in water. In this case there is required in the chemical reaction only about one part of water to thirteen parts to produce the 33 per cent acid, hence it will be noted that the quantity of steam blown in to form such acid is thirteen times greater than that which takes place in the reaction. The amount of steam, if the heated zone in the furnace be long enough, can be so reduced as to still permit much water being admitted to the coke towers to complete the condensation of all the acid.

Above, and on the end of the furnace where the cars are pushed in, is a large exhaust fan which draws all the gases from the furnace and forces them onward through a condensing system where the hydrochloric acid is condensed. The furnace is 184 ft. long, or ignoring the vestibule is 170 ft. long. Beyond the zone of high temperature towards the fan there is a distance of 79 ft. where the briquet-loaded cars act as a scrubber to remove any sublimed salt and other dust from the gases at the same time that the incoming briquets are being heated by the hot gases passing over them.

In addition to this we are constructing a dust-settling and cooling chamber in the iron pipe line between the furnace and the acid condensing system.

The reaction that takes place in the furnace will vary according to the composition of the aluminous material and the amount of alkali chloride mixed therewith, but typically it may be written as follows:

$$Al_2O_3\,2SiO_2 + 4NaCl + 2H_2O + yH_2O + zC + zO = (Na_2O)_2\,(SiO_2)_2\,Al_2O_3 + 4HCl + yH_2O + zCO.$$

The gaseous products in the above are carried into the condensing system after cooling. The briquets maintain their original form, and before the cars issue from the discharge end of the furnace they are cooled down by the inflow of the measured quantity of steam and air that is sucked in. This steam and air become preheated by the hot bricks before it reaches the combustion zone of the furnace, and the finished bricks are cool enough for men to handle them with gloves as they discharge from the furnace.

In our work at Sewaren we have discovered that our acid condensing apparatus with air cooling has only about one-third the capacity that it should have. It is at the present time being enlarged, and tourells submerged in water being added to it into which about 18 Beaumé acid will be run against the flow of the gases before the latter reach small coke towers. The escape of acid fumes has been so great that we have only been able to make four experimental runs of about thirty hours' duration each with the large furnace, but the results secured have taught us the necessary changes to be made in the acid condensation system and the necessary changes to get a full conversion in the furnace. The whole product of the last two runs has been analyzed, but not of the first two. The furnace was operated with only one set of oil burners and a car of briquets fed into the furnace during every forty minutes.

Before starting to build our plant at Sewaren briquets had been made containing as high as 33 per cent of sodic oxide and were white and free from iron.

We found by the analysis that the Sewaren briquets were but little more than half converted, hence it was rather against my desire to read this paper at this time. The briquets still contain a large amount of salt. The furnace is provided with another set of oil burners to meet this contingency, but we have not yet used them.

After completing the enlargement of our acid condensing system we contemplate using both sets of oil burners and adding a trifle more sawdust to the bricks, in which case there is no doubt but that we will produce the same results as in our laboratory work, with one exception. In the laboratory work the furnace was a deep, circular furnace filled with briquets and an oil flame and vapor of water were passed downward through the briquets. Under these conditions the iron as an impurity in the clay was carried downward as chloride of iron and small portions of it passed off with the acid and other portions condensed in the bottom of the furnace and in the iron pipes through which were drawn the gaseous product from the furnace from under a grate at its bottom.

As the bricks were secured white, it was concluded that we would secure the same result in our large work, but this conclusion, as given in a paper read before the Eighth International Congress of Applied Chemistry (this journal, Vol. X, p. 659, October, 1912) has, by practice, proven erroneous. As the surfaces of the cars are cooler than the briquets, we find a portion of the iron as either ferric or magnetic oxide on the car surfaces and a part still remaining mainly on the surface of the lower portions of the briquets. Thus it is the iron oxide remains on the cars and briquets and does not pass out with the gases to vitiate a part of the acid. The acid produced thus far only shows the presence of sulphuric acid by the most delicate test. It is free from arsenic and has but traces of free chlorine, iron chloride, sulphuric acid and ammonium chloride, and with our final settling dust chamber completed, it will be practically free from salt and aluminium chloride.

Lime Opening Process

In order to secure from the sodic-silico-aluminate material of the briquets the sodic aluminate contained therein we intend to put down a kiln of the rotary type to sinter this material, which is easily friable, after mixing with it lime in such proportions as to form di-calcium-silicate from the silica in their composition. This compound is insoluble, while the sodium aluminate is soluble. The material as thus treated is not fused, but sintered in such condition as to form an easily leachable mass. The reaction involved in this process is as follows:

$$(Na_2O)_2\,(SiO_2)_2\,Al_2O_3 + 4CaO = 2((CaO)_2\,SiO_2) + (Na_2O)_2 + Al_2O_3.$$

Limestone, calcite or marl in their chemical equivalents may be used instead of anhydrouslime.

The product thus secured will be leached by ordinary methods.

In the foregoing reaction any amount over two molecular weights of calcium oxide to each molecular weight of silica leads to the formation of insoluble calcic aluminate, and less than two molecular weights is not sufficient and leaves undecomposed a portion of the insoluble sodic-silico-aluminate. Under either of these contingencies, marked loss in the recovery of alumina may occur. Experimentally up to 90 per cent of the sodic aluminate has been leached from the above clinker, while there was left behind a mass of di-calcium silicate.

Production of Hydraulic Cement

According to Newbury's formula hydraulic cement is tri-calcium-silicate with up to 8 per cent of aluminium and iron oxides in a mixed condition being permissible in the formula. The above-mentioned di-calcium-silicate product is hydrated. If it now be ground and then heated with one added molecular weight of lime it forms a product that is in composition the same as that of hydraulic cement, but unless the material used is very pure, experience at Sewaren discloses that it will give a light gray cement and not a white, as the writer believed when he read a paper before the Eighth International Congress of Applied Chemistry.

Inadvertently there was printed in the paper read at that time the following statement:

"When briquets are made containing 32 per cent of sodic oxide the output of 30 per cent hydrochloric acid should be about 1½ tons of acid to 1 ton of salt and clay in the charge."

The statement should have read: To 1 ton of salt in the charge, about 1.75 tons of 33 per cent acid.

Use of Di-Calcium-Silicate as a Material for Glass Making.

If in the foregoing lime sintering process should there intentionally be left in the insoluble di-calcium-silicate after leaching 15 or 20 per cent of its alumina and alkali contents, there remains a product that is excellent for the manufacture of window and bottle glasses. Its use would call for the addition of silica and more alkali to its composition. Glass can stand a considerable percentage of ferric oxide and alumina in its manufacture.

OXALIC ACID

Nearly all the ingredients needed to produce oxalic acid, excepting sawdust, come from the products of these processes. The carbonic oxide from the acid towers can be scrubbed to free it from the last traces of hydrochloric acid and then passed over or through sawdust mixed with caustic alkali heated to the proper temperature producing an oxalate of the alkali, which treated with calcium chloride gives calcium oxalate and with sulphuric acid gives oxalic acid with calcium sulphate precipitated, frees the oxalic acid. This acid is most largely manufactured in Europe, but has a great consumption in America in our laundries.

USE OF HYDROCHLORIC ACID IN PRODUCING PHOSPHATE FERTILIZERS

The question may well be asked what use can be made of the large amount of hydrochloric acid that this process will produce should it supply as much alumina as will be needed for the production of the world's future supply of the metal aluminium. The author has pending a number of United States and foreign patents which may place hydrochloric acid in competition with sulphuric acid as now used in the manufacture of soluble forms of calcium phosphate mixed with calcium sulphate so largely used as a fertilizer. A pure calcium phosphate rock is anhydrous tri-calcium-phosphate. With it there is almost always associated considerable calcium carbonate, silica, alumina, and other impurities.

If this rock be ground to an even degree of fineness and then thoroughly digested with weak hydrochloric acid, the quantity of acid employed being theoretically just sufficient to dissolve all the free calcium carbonate, ferric oxide and one molecular weight of calcium oxide contained in the tri-calcium phosphate, the insoluble residue that is left is di-calcium phosphate. The soluble products, calcium, magnesium, and iron chloride, will exist in solution. By filtering and passing water through this mass, the soluble chlorides formed can easily be removed and either run into the sea or submitted to other possible uses.

The insoluble di-calcium-phosphate remains in a non-hydroscopic pulverent condition that can readily be applied to the soil by present methods of distribution. This is the product that all soluble phosphates revert to when they are applied to the soil as fertilizers. It has, however, one advantage over the soluble phosphates produced with sulphuric acid, namely, it is not loaded down with a large burden of calcium phosphate upon which transportation charges must be paid. If calcium sulphate is desired for fertilizer use it can more cheaply be produced from native gypsum rock. If the digestion is not thorough some soluble phosphates pass into the rich solution of chlorides. These phosphates can be precipitated by the use of a proper amount of calcium carbonate.

In Idaho, Utah, Montana, and Wyoming the United States Geological Survey have in recent years discovered and examined the largest deposits of phosphate-bearing rock ever known to the world. In this general region these rocks that have now been withdrawn from location covered an area as of Sept. 1, 1912, of nearly 2,212,000 acres.

The process as thus far described ends in securing the alumina as a sodic aluminate solution which can be completely purified from silica, titanic acid, and the iron oxide by a very simple means of precipitation. The final precipitation of the alumina in its purest form is brought about by Dr. K. H. Bayer's process, patented about 1894. It involves a very peculiar and interesting phenomenon. When previously precipitated alumina is added to the sodic aluminate solution even without shaking; if one observes a beaker containing this solution and the precipitate he will note the precipitate to begin to grow almost like a mushroom and in a few days' time nearly all of the alumina precipitates itself leaving a liquid containing nothing but caustic soda and a comparatively small amount of alumina. By an added step, which on account of patent matters I ought not to discuss, the last traces of alumina from the solution may be recovered and the solution left as strong solution of caustic soda needing slight concentration to prepare it for the market.

This general process opens the way to secure alumina cheaply or as a by-product in conjunction with these other products of great consumption in the industrial world. There is required 1.9 lb. of alumina to produce 1 lb. of aluminium, there being no appreciable loss in the present method of manufacture of the metal over the theoretical amount. This weight of alumina costs from 6½ to 7½ cents according to the latest data I have regarding the same and is the heaviest cost in producing the aluminium. By a general introduction of this process we may hope to see the cost of production of aluminium reduced 4 or 5 cents per pound, and when it comes to a general electrification of our world's railroads and a greater use of electric transmission lines and trolley wires, we will find aluminium a much more serious competitor of copper than at present, and eventually it will almost entirely supplant copper except in motors and generators.

POTASH ALUM FROM POTASH FELDSPAR AND ROCK PHOSPHATE

We all know the great need of potash salts as plant food and the scarcity of their supply, also the abundance of potash feldspar rocks containing great quantities of potash and the abundance of rock phosphate. Several years ago I was experimenting with the mineral wavellite, which is a hydrated phosphate of aluminium. Knowing that calcium sulphate or burned gypsum was slightly soluble in water, and that aluminium sulphate was very soluble in water, I reasoned that if proper proportions of aluminium phosphate and calcium phosphate were ground together and a minute amount of sulphuric acid were added to start the reaction, that the two bases should trade their acids if enough acid were present to form an insoluble form of calcium phosphate.

On trying the experiment, I found this reversal occurred and that aluminium sulphate could abundantly be leached away from the mixture. Further, knowing that di-calcium silicate was quite an insoluble body and if sulphuric acid were added to it the calcium sulphate formed would be but slightly soluble, it occurred to me that were I to take potash feldspar, for instance, orthoclase, whose formula may be written $K_2O\,Al_2O_3\,6SiO_2$, and grind it with just sufficient rock phosphate so that there would be present in the mixture two molecular weights of lime to each molecular weight of silica in the feldspar, or other potash silicates, such as leucite, and that if I added just the right proportion of sulphuric acid to this mixture I would have a chemical system which under conditions of solubility would degredate to the form of insoluble di-calcium silicate and soluble aluminium sulphate, potash sulphate and ortho-phosphoric acid, $H_3P_2O_4$.

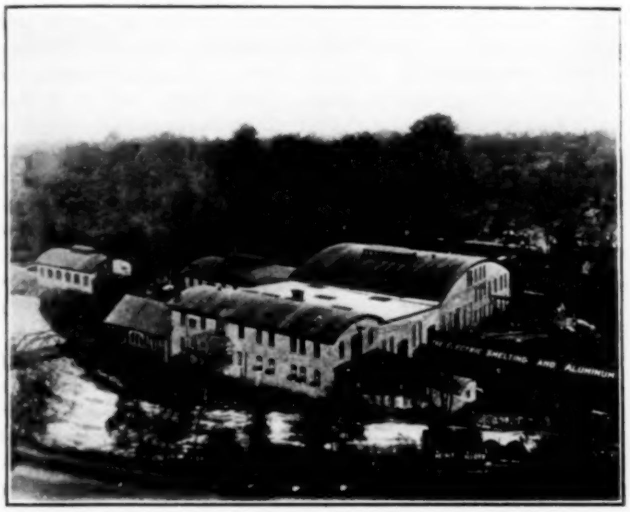
Upon having heated together the above mixture to merely a sintering temperature of about 1000 deg. C., and leaching with sulphuric acid, my first experiments gave almost 100 per cent.

For the purpose of this lecture I have had the work repeated. In figuring the amount of sulphuric acid to use, my assistant has taken a little less acid than was used in the former work and the result of the leaching has given a product containing all the potash and all the alumina from the feldspar, but only part of the phosphoric acid. Duplicate results have been secured substituting hydrochloric acid for sulphuric acid. The liquor in these two cases will give us either potash alum or a double chloride of aluminium and potassium associated with all the phosphoric acid contained in the rock phosphate employed. By this process the di-calcium-silicate or a slightly lower form of calcium silicate would be available for use in either the manufacture of cement or the manufacture of glass. Feldspar can be secured very free from iron, so also can rock phosphate. The iron, however, follows the other products into solution. From either potash, alum or the double chloride of aluminium and potassium the chances of securing pure alumina are very favorable. Upon heating to a dull red heat, aluminium sulphate breaks down into alumina with liberations of sulphuric acid (SO_3) and the potassium sulphate can then be leached away. In the case of a double chloride of aluminium and potassium at a very low heat in the presence of vapor of water, hydrochloric acid is given off with the formation of alumina, the chloride of potassium remaining as such, which can be leached away from the alumina.

To secure pure alumina from the above solution of chloride or sulphate salts with phosphoric acid is promising. The potash alum, however, cannot be separated by crystallization. By precipitation with sodic or potassic aluminate, if too much of either of these salts be not used, pure phosphate of aluminium may be secured. This audience will appreciate other possibilities.

POTASH PHOSPHATE FERTILIZER

In the foregoing I have described the sintering at a comparatively low temperature of rock phosphate and potash silicates. I wish to call attention to the fact that this mixture in



LOCKPORT PLANT OF ELECTRIC SMELTING AND ALUMINIUM COMPANY

some cases, and sinter in this instance with a proper amount of sulphuric acid added to it, or hydrochloric acid, if the hydroscopic properties be not detrimental in the latter case, is of greater fertilizer value than rock phosphate when used alone. By this new process the silica has taken place of much acid that would be required, and the product is not loaded down with calcium sulphate, but, instead, is loaded with potassium sulphate and the lime and silica is in such form as to be sufficiently soluble to furnish silica to plant food, and, further, the soluble lime is in a condition to act on potash minerals in the soil, displacing potash therefrom, thus rendering more potash available for plant food.

The soap industry is founded on the use of alkalis, and I have pointed out to you the possibilities of the fertilizer industry, the glass industry, and the cement industry as all being possible future contributors to the cheap production of aluminium. For the production of cheaper aluminium must come with the expansion of electricity to furnish power for the electrification of our railroads and other industrial uses.

Mr. Cowles then showed in the form of lantern slides the views of his Sewaren plant, which were given in our Vol. X, p. 659 (October, 1912), and also a view of his Lockport plant, which is herewith reproduced.

Mr. Cowles' paper was discussed at some length by various speakers.

The last paper on the program related to new uses of alumina, the author being Mr. L. E. Saunders, of the Norton Company, of Niagara Falls, N. Y.

Uses of Alumina as an Abrasive and Refractory

By L. E. Saunders

Most people undoubtely consider aluminium metal manufacture as the only significant factor in the consumption of bauxite and other aluminous ores, and the refined product alumina. I am therefore going to call attention to another industry which places dependence on aluminous ores, the abrasive business. Unless something calamitous happens to all of us during the next few months, this year will see a consumption of aluminous ores in the world corresponding to some 10,000 tons of aluminium metal in the manufacture of grinding wheels. It may not be realized to what extent abrasives enter in the equipment of the modern machine shop. Hardly an essential in our daily walks of life but is dependent on this new machine tool for its manufacture. The breakfast food with which we start the day, the stove on which it is prepared, the morning paper which gives it flavor, the street car, railroad train or motor car which takes us to business, the marble panels in our palatial offices, the typewriters which write our letters, the pearl buttons on our gloves, the leather gloves themselves, nearly all our activities, and finally, the polished granite columns at our graves would not be here in their present perfection were it not for the artificial abrasive which either makes an integral part or a machine to accomplish the end in view.

Broadly speaking, grinding is done in two classes: first, metals of high tensile strength, such as steel of all classes, malleable iron, etc., and, second, metals of low tensile strength, such as cast iron, brass, aluminium, etc. As about 75 per cent of the world's pig iron is converted into steel, and finished and wearing parts are largely made of this material, it may be seen that the first class covers the wider field. Aluminous artificial abrasives are used almost exclusively for this purpose and this fact offers a reason for the figures given above. The change from natural to artificial aluminous abrasives is world-wide, the use of emery and corundum showing a steady decrease. The reason for this lies in the lack of uniformity and grinding efficiency of the natural minerals and the possibility of exact duplication of quality in the electric furnace product. It is also possible to give such products varying characteristics for various uses, the accomplishment of which depends on the skill of the manufacturer and knowledge of the laws governing its properties. Such materials are therefore in use varying from an alumina content of 70 per cent to one of over 99 per cent.

As an adjunct to and an offshoot of this industry the aluminous refractory is coming into its own. (This journal, Vol. IX, p. 257, 1911.) While these articles are of little account as consumers of alumina, they are of considerable interest and value to the laboratory worker. The demands are increasing steadily as the users become better acquainted with the properties. One of the problems we have to contend with is to persuade the chemist that we have not attempted to furnish a sovereign remedy for all laboratory ills. The products may be varied in their properties, and an advantage in one direction is usually accomplished by the sacrifice of some other quality. Numerous published criticisms have appeared, most of which would have been quite uncalled for if the problems had been submitted first to the manufacturer so that a selection of the proper grade could have been made—neither in the abrasive or refractory field is alundum a panacea. When we are told the job, it is not very often that we fail.

In closing, let me say that new processes for purifying aluminous materials are still of great interest to us. A chance of securing cheaper raw materials looks good to us and from the user's standpoint we wish each experimenter success in his own work.

Industrial fellowships are established at the Universities of Kansas and Pittsburgh, under the direction of Dr. Robert K. Duncan, for the purpose of promoting the increase of scientific knowledge in chemical industries. The scheme involves financial assistance from any industrial concern with a problem to investigate, the co-operation of the university and the personal work of one or more fellows. The results, if any, are shared by the company, the university and the fellow. According to a recent announcement by Dr. Duncan before the Franklin Institute, the scheme is working well and has developed no weaknesses during the five years of its trial. Eleven fellowships have been completed at the University of Kansas and nine are still in force. At the University of Pittsburgh there are sixteen in force. The subjects of investigation have a wide range in industrial chemistry and metallurgy.

Titanium as Used in Steel Making

By E. F. Lake

During the last decade a great deal of progress has been made in the iron and steel industry and its products greatly improved by the perfection of new methods and the discovery and adoption of different elements for use as purifying or alloying material. One of the last elements in this category, but by no means the least, is titanium, which was first reduced to a fused condition by Moissan in his electric furnace. Although he separated it from all other elements he could not get it quite free from carbon. Between that time and the present, much experimental work has been done on metallic titanium but only very recently has it been made to better the various metal products when used in their manufacture. It was not until 1907 that the first heat of titanium steel was teemed and rolled in America and yet more tons of it are now made than of all the other alloy steels put together.

Metallic titanium is hard and brittle and has a dark grey color with a decided metallic luster; it being not unlike silicon and also resembling iron in appearance. It melts at about 2300 deg. C., and its atomic weight is 48.1. In nature it is never found native. It either occurs as titanium dioxide, (TiO_2), in the three distinct crystallized varieties of rutile, brookite, and anatase; in combination with iron, as titaniferous iron, containing ferrous titanate, ($FeTiO_3$), which is of the most interest to steel makers and sometimes contains 40 per cent of titanium, and in a number of iron ores and minerals. It is found in combination with the protoxide of iron, mixed with more or less of the peroxide of the same metal. It has been found in many minerals and rocks, as well as in clays and soils resulting from their decomposition. Thus it appears to be very widely distributed in the earth's crust but no one locality holds any large quantity. It has also been detected in meteorites and in the sun.

Titanium belongs to the same chemical group as silicon, zirconium, cerium, and thorium, but it occupies a position nearly midway between tin and silicon and by some might be grouped with the former element. In other ways it is allied to iron, chromium, and aluminium. It is attacked by common mineral acids and has a great affinity for both oxygen and nitrogen. Titanium burns more energetically in oxygen than any known substance and when heated in it creates an instantaneous dazzling flame like lightning. Its combination with nitrogen gas is attended with the evolution of heat. Moissan's investigations led him to the conclusion that it was the only undisputed example of the combustion of an element in nitrogen; while Woehler stated that no other element burns in nitrogen with so much energy as titanium. No other facts have since been brought to light to disprove these statements.

The titaniferous iron ores are very abundant throughout the world and in some parts exist in very large deposits. In fact whole mountains of it are located in the United States and Canada, and in Sweden and Norway. These were considered worthless for many years as the titanic acid produced a very infusible slag and attacked the lining of the blast furnace. It was even considered dangerous to the operations of the furnace to have too much of this ore present. In many cases the titanium combined with the nitrogen, set free by combustion, in the high heat of the blast furnace and formed a compound that entered the slag and made it so heavy it completely choked the furnace. Violent boiling has been caused in experiments carried on in the electric furnace and in some cases the charge has risen completely out of the furnace.

Most of this is due to improper fluxing, as the best of results were obtained in a blast furnace in the Adirondack Mountains in New York State, U. S. A., where iron ores very high in titanium were smelted for a period of something like twenty years. In England, at Norton-on-Tyne, the same thing has been done for several years. At the present time many tons of high-grade pig iron are being made from such ores in several different countries. They are generally so low in phosphorus and sulphur that ways and means are continually being found to use more and more of these ores. Many blast furnaces, that do not use titaniferous ores entirely, are mixing them with other ores for this reason. Another use to which they can be put is, as an addition to the bath in basic open-hearth furnaces. Here they would act as a desulphurizer and flux for the basic slag in place of fluor-spar as they perform exactly the same service as this latter material.

The greatest benefit that has accrued to the steel maker's art from titanium is not in the ways and means that have been devised for using the titaniferous ores, but from using the titanium in the ferro form, as a purifying and alloying material.

No method was found for reducing the ore and extracting the pure titanium until the electric furnace, with its high temperatures, was made a practical success. This was due to its comparatively high melting point of 2300 deg. C. Titanium also has a specific gravity of 5.174, which is considerably lower than that of iron at 7.2. This, in combination with the high point of fusion, made it impossible to combine metallic titanium with the molten bath of iron or steel, when held at workable temperatures, and hence no beneficial results could be obtained until ferro-titanium was produced and experimented with.

The nature and action of the metal was such that it encouraged investigation and experimentation. Through these it was found that raising the titanium content of the ferro alloy multiplied the difficulty encountered when trying to induce it to combine with molten iron or steel, and made impossible the task of getting it to enter into solution. On the other hand it was necessary to get enough titanium into the bath to perform the intended mission. The many tests made conclusively proved that the best results were obtained from ferrotitanium that contained from 10 to 15 per cent of titanic acid. When 20 per cent was present it would not dissolve thoroughly at the usual working temperature of an iron or steel bath, and higher percentages caused a segregation that produced hard spots in the metal. The 10 to 15 per cent ferro-titanium generally contains about 6 per cent of carbon and 5 per cent of all other impurities. As one-half of 1 per cent of titanium is all that is usually added to the molten steel, the percentage of carbon and other impurities, in the finished product, can easily be controlled.

The great affinity of titanium for both oxygen and nitrogen makes it of special interest to the steel maker and also to the steel user. It has been proven, beyond a doubt, that these two latter elements lower the static strengths and dynamic properties of steel, the wearing qualities and resistance to friction, and increase its tendency to corrode. That they are more injurious to steel than sulphur and phosphorus and present in larger quantities than has hitherto been supposed, is also considered a fact by those that have made the most careful investigations along this line. That a sulphur content as high as 0.13 did not develop "hot-shortness" in steel, and had no other injurious effects, was fully demonstrated by one series of tests in which the other impurities in the metal were reduced to traces only.

Removing the gases from steels, by the addition of titanium, directly increases the cohesive force in the mass by removing the microscopic, or even larger, bubbles which they form. These destroy the cohesive force between the molecules surrounding such spaces. Indirectly it strengthens the metal by preventing these gases from forming injurious compounds with other elements that are present.

As both oxygen and nitrogen are gaseous by nature it is difficult to analyze steels for these elements, but satisfactory results are now being obtained. In one lot of twenty-four specimens the oxygen content was found to range from 0.0[illegible] to 0.046 per cent. These may seem small enough to be ignored, but it is the combinations the oxygen will form, and not alone the percentage of this element, that must be considered when its effect on the steel is judged. As an example of this, we talk about the injurious effects of 0.05 per cent of sulphur, when it is really the 0.13 per cent of manganese sulphide that alters the quality of steel. Oxygen has

only one-half of the atomic weight of sulphur and, owing to its being capable of forming many more compounds, it exerts a much greater influence. Thus 0.05 per cent of oxygen may mean that 0.22 per cent of ferrous oxide is present and this is a large enough percentage to have an important influence on the metal. The oxygen was determined by heating very fine drillings of steel in a current of pure dry hydrogen, and weighing the water, formed by the reduction of the oxide.

When oxygen comes in contact with steel, either in a moist or heated atmosphere, oxides are formed. These show as tiny black specks under the microscope, when the surface is highly specular and magnified at least 1000 diameters. These oxides are always found in steels that develop blisters when being pickled. Bessemer steels contain the largest amount, with open-hearth steels second; crucible steels next, and electric furnace steels the most free from oxides. In fact steels have been made in the electric furnace that are apparently entirely free from oxides. The blisters are probably formed by the reduction of the oxides by the nascent hydrogen evolved during the pickling operation. When pickling high carbon steel rods they sometimes fracture in the bath and the same force that forms blisters on the softer steels might cause the harder steels to rupture. When titanium is added, in the proper manner, to the bath of molten steel it forms, with the oxygen, a titanium-oxide. This, being lighter than steel, floats to the top of the bath; enters the slag and is removed therewith. Thus no injurious compounds are formed with the oxygen.

That the reduction of the oxides is brought about by the nascent hydrogen may be easily demonstrated. To do this a polished surface should be made the negative electrode in a solution that is just strong enough to make it an electrolyte. The solution may be composed of 0.002 per cent, or two parts of a neutral salt, or an alkali, in 100,000 parts of distilled water. This should be heated to boiling and an electric current passed through it, to evolve the hydrogen in contact with the polished surface. After exposing the specimen for a few minutes it should be withdrawn and dipped in alcohol to prevent oxidation before it is examined.

Such a specimen will show that the manganese sulphide is unaltered, and the oxide is removed. The pits left by this removal of oxide will be larger than the specks of oxide formerly seen, as any oxide that may be in solution around these specks will also be removed. This explains why more oxide is found on chemical analysis than can be seen under the microscope. That both the current and the hydrogen are necessary can be proven by the fact that no pitting is produced by the same process minus the electric current.

That oxygen increases the tendency of steels to corrode is shown by the fact that polished specimens, containing oxides, begin to rust much sooner than those that are practically free from them. In dilute acid solutions this is greatly accelerated. That all impurities except oxide are electro-negative, as compared with iron, is generally accepted as the reason for the more rapid corrosion. Oxide of iron being electropositive when compared to iron, the electromotive force causes a dissolution of the steel in the vicinity of the oxide. This action continues until the oxide is removed by other means as, of itself, it does not dissolve with the surrounding steel. Welds are never perfect as we are not able to squeeze all of the oxide out of the joint. It there forms into small globules that interrupt the cohesive force which binds the molecules together.

These, as well as other, instances of the injurious compounds that are formed in steel by oxygen, show the necessity of removing as much of this element from steel as possible. While silicon, manganese, and other of the more common elements have been used for a long time for this purpose by steel makers, they still leave a percentage in the metal that is large enough to become an important factor. Quite recently, therefore, aluminium, vanadium, titanium, zirconium, etc., have been brought into use to remove the smaller percentage that is left by the ordinary steel making methods, and very good results have been obtained with all of them. Titanium does this work as effectively as any of the foregoing and is destined to be much more universally used as it is easily made to perform its mission and is much cheaper in cost. In the manufacture of steel rails with the Bessemer process it only adds about $1.20 per ton to their cost. In higher grades of steel the additional cost is seldom up to $2.00 per ton.

That nitrogen is injurious to steel is shown by the fact that each increase causes the elongation to rapidly diminish and reduce its ductility. At first a slight increase is caused in the toughness but after that this also rapidly diminishes, together with the loss of ductility and elongation. Generally speaking, steels have lost their elasticity, and their elongation and contraction have become nil when a nitrogen content of between 0.030 per cent and 0.035 per cent is in the steel. An 0.5 per cent carbon steel might not lose its ductility, however, until from 0.04 to 0.05 per cent of nitrogen was reached, and in the softer steels from 0.05 to 0.065 per cent might be present.

It might be claimed by some that hydrogen gas caused this brittleness that is assigned to nitrogen, or at least a part of it, as hydrogen imparts extreme hardness to otherwise pure iron. Hydrogen gas, however, is easily driven off by any of the reheatings the iron or steel gets and doubtless none of it can be found in the finished product.

In open-hearth steels the nitrogen content is usually between 0.020 and 0.025 per cent; in Bessemer steels from 0.018 to 0.062 per cent; and crucible steels from 0.010 to 0.015 per cent. Thus at least 0.012 per cent of nitrogen must generally be reckoned with. Steels made in the resistance electric furnace are practically free from nitrogen and hence are an exception to the above rule. Those, however, that are made in the arc electric furnace, in the presence of basic slags, are liable to contain enough nitrogen to be injurious.

The Manchester Steam Users' Association made some investigations of boiler plates that had failed. They found that the nitrogen content was from 0.0146 to 0.023 per cent and that this was from three to five times as great as the maximum found in any of the good plates that were tested. The notorious boiler plate from the Imperial Russian yacht Livadia contained 0.0123 per cent of nitrogen and 0.047 per cent of phosphorus. In the above tests a sample cut from a plate that burst under the hydraulic test showed 0.020 per cent of nitrogen and 0.052 per cent of phosphorus. Here the nitrogen was four times the amount of that found in any good plate tested. From these results the conclusion was drawn that nitrogen was more injurious to steel than phosphorus and that the one was as liable to produce brittleness and "cold shortness" as the other. It was concluded that no investigation was complete that did not determine the percentage of both phosphorus and nitrogen. They also established a rule as follows: The sum of the percentage of phosphorus, plus five times the percentage of nitrogen, should not exceed 0.08 per cent.

Nitrogen will not combine with steel by heating the two together, but in an atmosphere of ammonia they will unite. If the coking of the fuel has not entirely removed the nitrogen in the coal, ammonia is doubtless introduced into the blast furnace in that way. When nitrogen has once entered the pig iron in the blast furnace it cannot be removed by any subsequent reheating and hence is found in the finished steel.

Titanium, however, has a great affinity for nitrogen and when properly added to a steel bath it forms with it stable nitride which is lighter than the molten steel and raises to the slag. This nitride shows as tiny red crystals under the microscope. The very energetic reaction of titanium and nitrogen takes place at a temperature of about 800 deg. C. Combinations that are of no value are formed, however, if the air is carelessly shut off during this reaction and thus the good effects of their union will be greatly lessened, if not completely nullified.

Titanium is about the only element that readily combines with the nitrogen in steel and carries it away to the slag. While both silicon and manganese reduce the oxides they have practically no affinity for nitrogen. Chromium burns in oxygen at 1700 deg. C., but does not combine with nitrogen any more than does silicon or manganese. Molybdenum, even in pow-

dered form, will not form into combinations with nitrogen at temperatures up to 1200 deg. C. Tungsten will not form a nitride with nitrogen. Nickel also has no effect on nitrogen. While vanadium is a great scavenger of oxygen, it only effects the nitrogen in steel to a very limited extent. This leaves titanium alone as the nitrogen scavenger.

The proper method of adding the ferro-titanium to the molten metal is of vital importance. When improperly added it passes into the slag without going through the steel to search out and seize the oxygen and nitrogen and carry them with it to the slag. Some have advocated putting the ferro-titanium into the furnace but this only comes from a misconception of its nature and action, as silicon, manganese or spiegeleisen can better do the rougher work of oxidizing the steel and are much cheaper materials to use. Furthermore, adding the titanium in the furnace might cause it to work off into the slag and allow the gases to re-enter the steel before teeming. Titanium should only be used for removing the last traces of these gases, after the ferro-silicon and manganese have done their work, and hence should be added in the ladle.

The ferro-titanium comes well broken and ready for use. Being lighter than iron, it will not sink and disseminate when added near the top. Therefore, it should be put in, a shovelful at a time, while the molten metal is flowing into the ladle. If the stream is made to strike the ladle at one side a swirling and churning action is given the molten mass in the ladle and any ferro-titanium that might be floating on the surface will be engulfed and disseminated through the mass where it is quickly dissolved and starts on its mission. It should never be preheated as this sometimes prevents a complete reaction in the steel. Some begin by adding 1 per cent of titanium and gradually reducing this until the beneficial results obtained begin to diminish from the maximum point obtained.

The steel in the ladle should never be above the normal temperature for good untreated steel, as the reaction set up by the titanium causes the temperature to rise to a certain extent. After adding the titanium the ladle should be allowed to stand from five to fifeen minutes so the impurities and slag, released by this reaction, will have time to raise to the top of the ladle and not segregate in the steel. It is very difficult to make steel makers realize this as no other material they use acts in this manner. They fear the molten metal will become chilled and hence want to pour it into ingots before the titanium has accomplished its purpose. After standing, however, the metal is in better condition for pouring than it was before. Owing to an accident, one ladle had to be held for twenty minutes after teeming and it was then found to be in better condition for pouring into ingots, than was a freshly teemed ladle of ordinary steel.

Virtually there is no segregation in steel ingots containing titanium. Segregation is caused by accumulations of phosphorus and sulphur combinations, with iron and manganese, carbides, etc., along the main axis of the ingot. Where the metal remains longest fluid, near the top, there will be the most segregation. Selective crystallization as the steel begins to harden and the lower specific gravity of these combinations are the two principal causes of this segregation.

The gases liberated from the steel as it cools and hardens, raise in vertical streams and carry up the specifically lightest elements. Selective crystallization entraps them as the metal cools and the top of the ingot will hold the most segregation. If these gases are made to combine with an element like titanium, which carries them into the slag, the steel will harden with practically no occlusion of gases. Then very little segregation can be present, as the force is removed which caused the specifically lighter impurities to rise and collect in the top of the ingot. Titanium steels are also practically free from blowholes and the above described action of titanium is doubtless the reason.

If there is any titanium in the steel after the occluded gases have been removed it has a tendency to attack the sulphur and phosphorus and either counteract their injurious effects or remove them. The sulphur is reacted on and carried away to the slag in the form of a sulphide, or sulpho-cyanide, of titanium. Special means have made the phosphorus pass into the slag as phosphate of titanium. Thus the mission of titanium does not seem completed after the oxides and nitrides have been removed.

Ferro-titanium has been used in all of the steel making processes, including the electric, and the metal has been made stronger and longer lived thereby. It was thought that the electric furnace steels would be made so free from gases of all kind that they would not need the service of any purifying element. This, however, has been proven erroneous, for titanium is now used for the removal of the last of the gases in electric furnace steels. It is less expensive and troublesome to use this material than to remove all of the occluded gases with the electric process. An exception must be made in the resistance electric furnace however, as steels made in that type are practically free from oxygen and nitrogen.

The abrasive or frictional wearing properties of titanium steel are well illustrated by the sections of 90-lb. rails which have recently been taken up after being in service for twenty months on the high-side of two curves, of practically the same curvature and grade, on the Lehigh Valley R. R. The average of a number of rails, measured at this same time, gave a ratio of wear of 1 to 1.54, in favor of the titanium treated rails. The deflection was 1.40 inches for the titanium and 1.56 inches for the ordinary rail.

All of the tests that have been given steels for endurance show that titanium greatly increases this property. In one test of open-hearth steel, on the White-Southers rotary vibrational machine, it withstood 2,660,000 revolutions before it broke. The same steel, treated with titanium, withstood 18,274,900 revolutions before it broke. Other tests such as impact, cold bend, alternating vibrational, compressive and tensile strength show an improvement when titanium has been properly added to the metal. Such encouraging results have been obtained with steels that a large amount of work is now being done on the brasses and bronzes so titanium can be successfully used in the non-ferrous field. It has been used for some time in copper and greatly reduces the blow-holes when casting this metal. Very good results have also been obtained when using it in iron and steel castings.

The production of gold, silver, copper, lead and zinc in the Western States and Alaska in 1911 has been announced by the United States Geological Survey in two advance chapters from Mineral Resources of the United States, 1911. Many interesting facts and figures are given showing the sources and distribution of the different metals. Milling processes produce over half the gold, while smelting produces over four-fifths of the silver. The report gives the ore tonnages and metal contents for the different districts in the various states. Copies of the reports may be had on application to the Director, United States Geological Survey, Washington, D. C.

The size of the feed to Chilean mills is an important factor in their operation and efficiency. According to Mr. G. A. Denny, *Informes y Memorias* del Inst. Mexicano de Minas y Metalurgia, Vol. III, No. 1, it is a wholly mistaken idea that a Chilean mill has a larger output if fed with fine material than with coarse. Contrariwise, it has a considerably higher output on ore fed from breakers set at 1.5 in. than on a pulp discharged from a 3-mesh screen, and probably four times the efficiency on the coarse feed cited than on ordinary middlings or tailings.

The effect of flotation on the world's supply of zinc was first noted in 1904, and in 1906 the spectacular rise in the production from Broken Hill began. Between 1906 and 1911 the production rose from 100,000 tons of concentrate to 500,000 tons. As a result of a statistical study of zinc production at Broken Hill Mr. T. J. Hoover is of the opinion that no further large zinc concentrating mills will be erected there; that there are no further large discoveries of new deposits or lodes in the district, and that the quantity of tailings now stacked is accurately known, as is the quantity being treated. He assumes that the production for 1913, 1914 and 1915 will be between 400,000 and 450,000 tons of concentrate per annum.

The Gayley Dry Blast Process*

By Prof. Henry M. Howe

Interesting as are the other aspects of the Gayley process, its greatest interest lies, I think, in the light which it throws on the nature of expert evidence and on the value of expert opinion.

What is this process? It is simply drying the air used for burning the fuel in the iron blast furnace, apparently a rather simple matter. To refresh your memory, the blast furnace is a huge vertical firebrick cylinder, roughly speaking. In it iron ore, which is essentially iron oxide, is converted into cast iron or pig iron by prolonged exposure to coke or its equivalent, in an atmosphere of mixed carbonic oxide, carbonic acid, and nitrogen, which results from burning this coke by means of atmospheric air forced in through appropriate openings near the bottom of the furnace, and called "the blast." The furnace is full from top to bottom of an intimate mixture of this coke and ore, together with limestone added for the purpose of forming with the barren mineral matter of the ore and with the ash of the coke a fusible silicate or slag, and for other purposes into which we need not enter here. These three solid materials collectively are called "the stock."

The burning of the coke generates at the bottom of the furnace a temperature so high as to melt away the bottom of this column of stock, or mixed coke ore and limestone, of which the last two have by this time been converted locally into metallic iron and lime, and as the column thus descends it is renewed at its top by adding more of this same mixture so as to keep the furnace continuously full.

There are, as it were, two rivers passing through this great cylinder in opposite directions, a sluggish river of solid stock which descends as its bottom is melted away, and a swift river of ascending gases resulting from the burning of the coke by the injected air or blast. The former traverses the length of the furnace in from 12 to 15 hours, the latter in a very few seconds. These rivers interact as they interpenetrate and flow past each other, the rising gaseous column progressively giving up its heat to the solid column and taking from that solid column the oxygen of its iron oxide, so that the gaseous stream as it emerges from the top of the furnace has taken from the descending ore all the oxygen that it is capable of removing, and has delivered over to that ore and its accompanying coke and limestone all the heat that they are capable of taking from it.

So much for the blast furnace process, which, like every mundane process that seems simple, is in fact of a complexity so overwhelming that the human mind is inherently and incurably impotent to grasp it. We rub our eyes and seeing as far as the ends of our noses, assume that we see to the end of the universe.

As a heat engine the blast furnace was known to be extremely efficient, as human heat engines go. Nevertheless by the extremely simple step of drying the blast Mr. Gayley made a further important saving of fuel amounting in some cases to 20 per cent, and according to our present evidence to about 10 per cent on an average, for usual American conditions. To those who had taken up the logical stick by the wrong end, such a saving by thus drying the blast seemed simply preposterous, as preposterous as talking a thousand miles through an iron wire, or waving a message to Europe without any wire, or as any invention is till you understand it.

Its preposterousness was promptly and convincingly exposed by the public spirited experts whose geographical misfortune prevented their knowing Mr. Gayley's character, though those of us more favored geographically followed "Br'er Fox" till our question "Why and how?" could be answered.

The heat required for heating and dissociating the moisture of the blast is a small fraction of the total heat requirement of the blast furnace process; how then dare we say that the removal of this heat requirement may save 10 or even 20 per cent of the fuel?

Not a few of us had grown so used to calculating the heat needed for a given chemical and physical work like that of the blast furnace, and the heat evolved by the combustion of a given weight of fuel under known conditions, and to calculating thus the thermal balance, that we had lost sight of the supreme importance of temperature.

The atmospheric moisture, calorimetrically considered, is indeed slight, but thermometrically considered it is sometimes of overwhelming importance. As to its effect on climate Tyndall says "To say that on a day of average humidity in England, the atmospheric vapor exerts 100 times the action of the air itself, would certainly be an under-statement of the fact." . . .

The sum and substance of it is that the blast furnace process has among its various duties to supply a certain quantity of heat at or above a certain high temperature, which for brevity we may call the critical temperature, while shamefacedly confessing that we are further overworking an already grossly overworked word. No quantity of heat offered at any lower temperature will do this work. All the heat in all the suns of the milky way, if offered at a temperature of 50 deg. C., could not boil one egg.

Drying the blast saves fuel by improving the temperature-distribution of the heat generated.

Perhaps this is more readily understood if we consider an imaginary case. Suppose that matters were so ordained that he who wished to build an iron building was obliged to take, along with his iron work, a certain fixed proportion of accessories, windows, flooring, tiling, etc., the proportion fitted for a ten-story building. That would be all well enough for those who build nine, ten, or eleven-story buildings; but for the builder of a forty-story building it would be most uneconomical, because whereas the quantity of accessories, windows, flooring, tiling, etc., increases directly as the height of the building, the iron work needed increases in a higher ratio; so that, in order to get enough iron work for his forty-story building he might have to get enough accessories for a sixty-story building, and the excess would be left on his hands. This is because of bad proportioning of his supplies. A mere change in the proportion between iron work and accessories would cause a great saving.

It is somewhat so with a thermal process like that of the iron blast furnace. When combustion has raised the temperature above the critical point there is available first a certain quantity of heat above that temperature, the hyper-critical heat, and second another and larger quantity of heat below that temperature, the hypo-critical heat, the heat remaining when the temperature has sunk to the critical point. The hyper-critical heat is needed for certain work which can be done only at hyper-critical temperatures, the hypo-critical heat is available for work which can be done at hypo-critical temperatures; so that, as we have hyper and hypo-critical heat, we have also hyper and hypo-critical work.

For heat economy the hyper-critical heat should be to the hypo-critical at least as the hyper-critical work is to the hypo-critical work. This is indeed true of every specific temperature covered by the blast furnace process. The proportion between the two evidently depends on the temperature reached by combustion itself. The higher this temperature, the higher proportion does the hyper-critical heat bear to the hypo-critical heat.

If the temperature developed by combustion is so low that the ratio of hyper to hypo-critical heat is too low, is below the ratio of hyper to hypo-critical work, then in generating enough hyper-critical heat to do the hyper-critical work we are forced to generate an excess of hypo-critical heat over and above the hypo-critical needs of the process, and this excess of hypo-critical heat is used to poor advantage or is wasted. If my ratio of iron work to accessories is too low, then in providing my building with enough iron work for forty stories I am forced to provide enough of these accessories for sixty stories, and the excess is left on my hands. And as a mere change in

*Extracts from the address made by Prof. Howe at the presentation of the Perkin medal to Dr. James Gayley at the meeting of the New York section of the Society of Chemical Industry on January 24, 1913. An account of the proceedings, together with Dr. Gaylty's address in full, was given in the February issue of this journal, page 71.

the ratio between iron work and accessories can enrich or ruin a builder, so a mere change in the ratio between hyper- and hypo-critical heat, induced by a change in combustion temperature, can lead to a wholly disproportionate change in the economy of our own vital processes or of the process of the blast furnace.

Thus it was with Neilson's introduction of the hot blast. Formerly blast furnaces were fed with cold air, with the result that, because of the low temperature of combustion, the proportion of hyper-critical to hypo-critical heat was far below the proportion of hyper- to hypo-critical work to be done, with the result that the burning of enough fuel to provide enough hyper-critical heat yielded a quantity of hypo-critical heat far in excess of the hypo-critical work to be done, and this excess was used to poor advantage or wasted. Hence raising the temperature of combustion by heating the blast led to a saving of fuel which, to those unable to think, was miraculous.

The degree of economy caused by blast-drying should vary from case to case with the initial lack of hyper-criticalness in the combustion temperature; and if there is no such lack initially, as may happen, then blast drying should cause no economy.

Other means of adjusting the ratio of hyper- to hypo-critical heat suggest themselves, such as raising the temperature of combustion by further preheating the blast, by enriching it in oxygen by removing part of the atmospheric nitrogen, or by electric induction at the very focus where the hyper-critical work goes on; and lowering the critical temperature by changes in the conduct of the process. We have not reached the end of knowledge in general, or of the improvement of the blast furnace process in particular. But the fact that blast drying, in removing the effects of the fluctuations in the atmospheric moisture removes a serious cause of irregularity in the working of the furnace and in the quality of its product, gives it an administrative and commercial advantage over other means of raising the combustion temperature which may well be decisive. . . .

The deliberateness and caution, indeed the tardiness, with which the iron trade has proceeded in adapting dry blast is referable to various reasons, such as hesitation to incur the certainly great expense of its installation, the competition of other devices for increasing earnings in other directions, the wish of each to get the benefit of the experience gained by its earlier adopters, uncertainty as to whether the saving found by John Doe will apply to the different conditions of Richard Roe's furnace, and the like.

But, after all is said and done, that which interests us most is not the invention itself, important as that is, nor the great saving of fuel. The striking thing is the contrast between the mental attitude of the certainly very learned men of science who immediately stamped Mr. Gayley's claims as preposterous, and the attitude of this great Captain of Industry, who not only saw the saving to be effected but saw it so clearly that he was able to bring to pass the very costly experiments needed to prove his faith.

Let us learn the lesson of humility. Natural human caution is likely to prevent the cautious from saying "I know that so and so can be done" unless they do know it; but such cases as this show that it does not prevent even the well qualified, the expert, and the prudent from saying "So and so can not be done," though in fact it ought to prevent them, in view of the almost infinite excess of our ignorance over our knowledge.

Columbia University,
New York City.

The Utah Society of Engineers has recently secured new quarters in the Salt Lake Stock and Mining Exchange. The business office is in room 21, and the regular monthly meetings will be held in the large exchange room. The removal of the Society from its small quarters in the Newhouse building was necessitated by its rapid growth in the last few months. Mr. R. K. Brown is president of the Society this year.

Ore-Testing at the Colorado School of Mines

The Board of Trustees of the Colorado School of Mines has approved a system of charges to be made for the use of the experimental ore dressing and metallurgical plant of the school. Communications for further information should be addressed to the manager, William G. Haldane, Golden, Colo.

PRELIMINARY TESTS

The equipment consists of one 24-in. Wilfley table, one 4-in. by 6-in. two-compartment jig, one set hydraulic classifiers, one 6-in. by 48-in. amalgamating plate and one set of cyanide agitators. Lots of 10 to 75 lb. of ore are suitable for these tests. Price for each lot tested, $25, including assays and analyses.

INTERMEDIATE TESTS

Small jigs and quarter-size Wilfley and Card tables. Lots of 100 to 1000 lb. of ore are desirable. Price for each lot tested, $75, including necessary assays and analyses.

CYANIDATION

Preliminary cyanide tests will be made on lots of 25 lb. of ore. Price for each lot tested, $30, including necessary assays and analyses. A detailed statement of results will be given, but the report will not embrace conclusions or advice. A consulting engineer should be employed to interpret the results and advise his client.

CARLOAD LOTS

In this work the full-size equipment of crushers, rolls, screens, jigs, regrinders, classifiers, tables and vanners may be used. These tests should be supervised by the customer's consulting engineer. The school does not assume responsibility for the results nor advise the adoption of any special treatment, but will place the appliances of the plant at the disposal of the consulting engineer. Charges will be from $10 to $100 per day for the use of such portion of the plant as needed.

DAILY CHARGES FOR USE OF PORTION OR ALL OF PLANT

Preparation and Sampling.—This includes weighing, crushing in Blake or Gates gyratory crushers, sampling by Vezin or Brunton samplers, and delivery of the ore to roll or stamp bins. Price, $50.

Concentration.—Rolls, elevators, trommels, jigs, regrinding machines, classifiers, sand pump, tables and vanners. Price, $50.

Amalgamation.—Stamp mill, with plate or inside amalgamation, classifiers, tables and vanners. Price, $40.

Cyanidation.—Dorr classifier, tube mill, Dorr thickener, sand leaching tank, mechanical agitation tank, Paterson circulation treatment tank, pumps for sand, slime, solution and vacuum, Moore filter, Butters filter, zinc boxes and lead-lined acid tanks. Price, $40.

Magnetic Separation.—Dings two-magnet separator with means of securing magnetic fields of high and low intensity. Price, $10.

Magnetic Roasting.—Wilfley annular hearth revolving furnace, with pyrometers for temperature control. Price, $15.

Electrostatic Separation.—Huff electrostatic separator, current-producing devices, and two types of separators for fine and coarse material. Price, $10.

Use of entire plant per day, $100.

ORE SHIPMENTS

Ore in small quantities may be shipped by Adams express to Golden, Colo. Larger lots should be sent by freight via Colorado & Southern Railway, or via Denver & Intermountain railway. All charges should be prepaid. Ore sent by freight must be hauled by wagon from the railroad to the plant at the shipper's expense. A deposit to cover all charges for ore treatment must be made in advance. When different tests are made on the same ore, only one charge will be made for assaying and analysis.

In gaging wire it is better to use a micrometer than either a disk or forked gage. The two latter are not as free from objections as the first. With a disk gage fine sizes of wire can be stretched in forcing the wire in the notch, while with forked gages it is possible to spring the gage to show wide variations. Neither the disk nor forked gage will give as close a measurement as the micrometer.

Annual Meeting of American Institute of Mining Engineers

Adoption of New Constitution—An Embarrassment of Riches in Professional Papers—The Symposium on Sound Steel Ingots

THE annual meeting of the American Institute of Mining Engineers, held in New York City on February 18 and 19, will go down into history not only as one of the most successful meetings from a professional engineering standpoint, but as the beginning of a new era in the Institute's history, as the result of the adoption of the new constitution.

On the evening of Monday, February 17, the members assembled for an informal social gathering at the Institute headquarters in the United Engineering Building. They listened to a gracious word of welcome by Dr. George F. Kunz and to a delightful address by the president, Professor Kemp, giving a sketch of the life work of the new president to be elected on the next day, Mr. Charles F. Rand, and enlivened by lantern slides in Professor Kemp's unique style which leaves the audience in constant reflection as to the extent of the genuineness of the photographs.

Business Meeting and Presidential Address

The morning session of Tuesday, February 18, was given over almost completely to the business meeting of the Institute. The chief result was the announcement that by the mail vote and the additional vote of those present the new constitution of the Institute has been adopted in the whole with an overwhelming majority, practically unanimously.

There was, however, an extra vote to be taken on four amendments by Messrs. Corning and Stone on minor matters, all of which were adopted, and on the provisional clause in the constitution providing for a special class of "Fellows" which should have the right to express an opinion with regard to pending or proposed legislation affecting the mining and metallurgical industries. The mail vote on the Fellows proposition showed a very slight majority against it (279 against the class of Fellows and 260 in favor of it). As this result could have easily been changed by the vote of those present, the Committee on Constitution, through Prof. J. W. Richards, explained that they had submitted the amendment concerning the class of Fellows by request and that they did not favor it now in view of the proposed union of the Mining and Metallurgical Society of America with the American Institute of Mining Engineers. Thereupon, the vote of those present was practically unanimous against the class of Fellows and a few who had voted by mail changed their votes, and the final joint result of the mail vote and of the votes of those present was 313 against the clause of Fellows and 258 for it.

Dr. Douglas in his warmly applauded report for the Land Committee stated that on the debt of $68,000 there had been subscribed already more than $59,000, and urged those members who had not yet contributed to make up the small balance. This result seems to make sure that it will not be necessary to raise the membership fees, as with the land debt paid off the fixed charges of the Institute will be reduced by $3,000 a year. Resolutions of thanks were voted to Dr. Douglas.

Charles F. Rand
President American Institute of Mining Engineers

The result of the election of officers was then announced as follows:

Mr. **Charles F. Rand,** president of the Spanish-American Iron Company, was elected president.

Messrs. F. W. Denton, Karl Eilers, Sidney J. Jennings, T. H. Leggett, Waldemar Lindgren and B. B. Thayer were elected vice-presidents.

Messrs. J. W. Finch, James Gayley, J. A. Holmes, R. W. Hunt, John H. Janeway, E. P. Mathewson, W. J. Olcott, J. W. Richards, L. D. Ricketts, C. S. Robinson and E. L. Young were elected directors.

By the new constitution the secretary has ceased to be elected by the members at large and is being appointed by the Board of Directors. The proxies which had been made out were, therefore, not voted. The Board of Directors held meetings on Tuesday and Wednesday and during the afternoon session of Wednesday the new president, Mr. Charles F. Rand, brought to the members a "message from the Board of Directors," announcing that after several hours of consideration the Board had selected Mr. **Bradley Stoughton** for secretary.

The morning session of Tuesday was concluded by the address of the retiring president, Professor Kemp, which dealt with the subject of the ground waters in a very interesting manner. Lunch was then served and the reading and discussion of papers were taken up in the afternoon session.

Tuesday Afternoon Session

The papers presented in the afternoon session of Tuesday related to geology, mining and allied subjects.

The Rôle of Certain Metallic Minerals in Precipitating Silver and Gold

A paper by Dr. **Chase Palmer** and Dr. **Edson S. Bastin,** of the U. S. Geological Survey, was presented by Dr. Bastin and illustrated by experiments. The authors have made extended laboratory experiments on the rôle which certain metallic minerals play in precipitating silver and gold from solution and apply their chemical results to draw some interesting geological conclusions.

Their experiments show that certain sulphides, arsenides and sulph-arsenides of copper, nickel and cobalt precipi-

tate metallic silver very efficiently from dilute aqueous solutions of silver sulphate. As the waters descending through the upper portions of most sulphide ore bodies are known to be sulphate waters, similar precipitative actions would be expected under natural conditions. The frequent association of silver in ore deposits with chalcocite and bornite, and particularly with niccolite and cobaltite, minerals which in the authors' experiments were among the most efficient precipitants, warrants the belief that such reactions are of great importance in secondary enrichment of ore bodies.

The more common sulphides such as pyrite, galena and sphalerite were relatively inactive as precipitants of silver from aqueous solutions of its sluphate.

The quantitative results obtained with niccolite and chalcocite indicate that the essential chemical changes in reactions of this type are due to oxidation through the hydrolitic action of water. It is apparent, therefore, that certain water solutions may act as potent oxidizing agents below the ground-water level.

The experiments indicated that nearly all of the sulphides and arsenides common in ore deposits were capable of reducing gold from a solution of its chloride, although important differences in the rapidity of the precipitation were observed with different minerals. Most of the minerals that are especially efficient as precipitants of silver are also effective precipitants of gold and a number of other minerals such as galena, pyrite, stibnite and millerite that are inefficient in precipitating silver are efficient in depositing gold.

It is known that the waters descending through the upper portions of sulphide ore bodies universally carry chlorides and it is probable that these chlorides have effected the solution of the gold. It is probable, therefore, that phenomena similar to those exhibited in the authors' experiments with gold chloride solution play an important part in secondary enrichment in gold.

While the phenomena described by the authors find their most immediate application in secondary enrichment it is perfectly possible that such reducing effects of the sulphides may be responsible in part for the primary association of the precious metals with certain sulphides in preference to others. It is recognized, of course, that certain mineral associations in ore deposits are probably the result of processes analogous to differentiation in rock magmas, but it is quite possible that other associations such as the apparent preference of gold for chalcopyrite and tetrahedrite rather than for pyrite in deposits carrying these three minerals may be due to differences in the reducing power of these sulphides themselves. Light could probably be thrown upon this point by the investigation of the reducing effect of various sulphides upon silver and gold salts dissolved in solutions having the composition of certain deep mine waters.

It is a generally recognized fact that the purity of alluvial gold is greater than that of the veins in the neighborhood. This superiority in fineness has generally been explained by the well-known fact that silver is more readily soluble in natural waters than gold, and is by them removed from the natural alloy, thus increasing its purity.

Mr. Lindgren has recently discussed (U. S. Geo. Survey, No. 73, 1911) this matter at some length in a report on the Tertiary Gravels of California, and has presented a large amount of statistical data leading to the same conclusion. It has been thought by certain geologists that this refinement of the gold was accomplished by solutions circulating through the gravels themselves, but Mr. Lindgren states that "so far as the tertiary gravels of California are concerned, the conclusion of the writer is that solution and precipitation of gold have played an absolutely insignificant part."

Under the conditions of the experiments reported by the present authors it was found that nearly all of the metallic minerals common in precious metal deposits were capable of precipitating gold while a much smaller number, and these not the most common ones, were active precipitants of silver. When it is remembered that the source of the placer gold is the oxidized zone of the original deposit and that the gold may have been dissolved and redeposited several times within the vein before erosion carried it into the alluvium, it seems not improbable that such selective precipitation may be an important factor in this natural refining of gold.

A Microscope Study of Sulphide Ores of Copper

A paper by Dr. **L. C. Gradon,** of New York, and Dr. **Joseph Murdoch,** of Harvard University, Cambridge, Mass., was entitled "The Sulphide Ores of Copper—Some Results of Microscopic Study." The paper gives a very large amount of facts, experimentally established, and is illustrated by a series of thirty-six microphotographs of striking significance.

In the introduction the authors discuss the relation of scale in geologic work and the general characteristics of copper sulphide ores and outline the scope of their investigation. Then follow descriptions of the important minerals—pyrite, pyrrhotite, chalcopyrite, bornite, covellite, chalcocite, enargite, tetrahedrite and tennantite. Each of these minerals is discussed as to primary character, alteration and secondary derivation.

Some of the more important conclusions are as follows:

Conclusion as to whether an ore or a given mineral in an ore is primary or secondary can be safely drawn in the great majority of cases from the type of structure involved.

In the case of chalcocite, internal or molecular structure as expressed in cleavage may also be used with caution as a criterion of the primary or secondary character of the mineral.

The following sulphide minerals common in copper ores are primary, i. e., they are products of original precipitation, probably from ascending, heated, alkaline solutions: pyrite, pyrrhotite, chalcopyrite, bornite, chalcocite, enargite, tetrahedrite, tennantite, spalerite, galena, and probably covellite.

The following minerals are secondary, i. e., they are derived by the action of descending cold acid waters upon the other sulphides, i. e., secondary sulphide enrichment, chalcopyrite, bornite, covellite, chalcocite and enargite.

The most important result of this study is perhaps "the abundant evidence afforded that primary copper deposits are definite, characteristic, clear-cut phenomena of striking similarity in the main, formed by well-defined processes as a result of certain specific causes acting through a limited and probably brief period of time. Among the ores that have been studied, at least, there are absolutely no indications favoring—rather, there is plentiful testimony against—the hypothesis that many or most ore bodies (excluding the enriched zone) are the result of 'repeated reworkings' of their constituents by various processes, of different causes, at separated times or over long extended periods. This idea, never ringing true nor supported by competent evidence, has served only to complicate and befog the situation, and thus postpone universal acceptance of the fast-growing and now almost overwhelming proof of the direct and sole influence of igneous agencies in the formation of many ore bodies."

The following papers were also presented during the Tuesday afternoon session by their authors:

A Problem in Mining together with Some Data on Tunnel Driving. By **F. M. Simonds** and **E. Z. Burns.**

The Geographical Distribution of Mining Development in the United States. By **E. W. Parker.**

Electric Power Installation at El Tigre, Sonora, Mexico. By **J. W. Malcolmson.** (This paper contains some interesting figures of cost.)

Structure of the Northern Anthracite Basin Relative to Forms of Folds. By **N. H. Darton.**

Among the papers which were on the program for the afternoon, but which were read by title only, are the following:

The Hardinge Conical Mill. By **H. W. Hardinge.**

Fire Clay Deposits in Canada. By **Heinrich Ries.**

School Laboratory Work; the Sampling of an Ore Containing Coarse Gold. By **Charles F. Locke.**

In the evening a very interesting illustrated lecture was given by Dr. **Fred. Haynes Newell,** director of the United States Reclamation Bureau, on "Immigration and Its Relation to the Mining Industry." This lecture was greatly enjoyed by a large audience.

Symposium on Sound Ingots

The morning session of Wednesday was devoted to the reading of five papers relating to methods for improving the soundness of steel ingots and to a general discussion of the subject. There were some 200 members and guests

BRADLEY STOUGHTON
Secretary American Institute of Mining Engineers.

present, and the attendance was truly representative of the industries interested in sounder ingots. But the representatives of most of the big steel companies showed a rather regrettable reluctance to express their views. Nevertheless, the meeting was exceedingly interesting and profitable.

This symposium had been arranged by the Iron and Steel Division of the Institute, with Mr. Charles Kirchhoff, chairman, and Mr. Bradley Stoughton, secretary. As mentioned before, Mr. Stoughton is now also secretary of the Institute as a whole.

Piping and Segregation of Ingots of Steel and Ductility Tests for Open-Hearth Steel Rails

A paper by **Dr. P. H. Dudley** of the New York Central Lines on the above subject was first presented. It was an elaborate discussion of extended tests made on this subject.

Mr. Dudley called attention to the fact in the specifications for the New York Central Lines that in mill practice as soon as the ingots are stripped they should be charged into the reheating furnaces to prevent the setting steel to cool from its molten temperature to that of cold metal, and thus avoid the formation of the full shrinkage cavities in the ingots. It has been shown by the cutting of a large number of blooms that it is possible to prevent a shrinkage cavity from forming of not more than from 1/20 to 1/30 of the size in the top of the hot ingot by this method of good mill practice, of what would be formed by permitting the ingot to become completely cold before it was put into the reheating furnace for rolling.

It is first necessary, however, to provide the hot molten metal with a chemical composition which will produce sound ingots and definite physical properties in the finished product. The first effort is to secure a well-deoxidized steel, and by proper mill practice make sound ingots.

Dr. Dudley does not consider it advisable to follow the suggestion of Mr. Talbot and Sir Robert Hadfield to use a large percentage of aluminium in the ingots to reduce more completely the oxides, in view of the difficulties already experienced with aluminium so used in rail steel for heavy wheel loads. "It would be better to use silicon or a combination of silicon and ferro-titanium to secure the desired results. We do not use as high percentages of silicon in steel as is employed abroad, except for tires."

Dr. Dudley then dealt at greater length with the segregation of basic open-hearth steel ingots, giving analytical diagrams of various ingots and finally discussed the drop and exhausted ductility tests for basic open-hearth steel rails.

Dr. Dudley's final conclusions for the manufacture of the present basic open-hearth rails are as follows:

"1. The chemical composition should provide for sound steel of ample physical properties of tenacity and toughness rather than hardness combined with brittleness.

"2. The impurities, phosphorus and sulphur, should be of minor content so the bath of metal can be purified to produce the large percentage of toughness and ductility due to the specified chemical composition.

"3. The ingot should have such relations of area of base compared to the height and weight that under good mill practice and suitable deoxidizers it can be made with controlled segregation, and only a trace of a shrinkage cavity in the top; then, when bloomed under its equalized initial heat, it is rendered pipeless by the usual 8 to 10 per cent discard.

"4. Aluminium can be replaced and silicon partly, as deoxidizers, with advantage by the use of ferro-titanium to purify, solidify and check segregation in rail, tire and axle steels, and also some of the lower grades of carbon steels where great purity is desired.

"5. The ductility and elongation tests to date furnish the best and only prompt means of determining the degree of purification of the steel per melt as it is made by indicating the physical properties secured before another melt is tapped from the same furnace, and are of decided advantage to the manufacturer as well as to the consumer. These tests are so advanced that they must be applied with knowledge and understanding for proper results, and not made mechanically for specified records.

"6. Every process or step of the entire manufacture of the steel, rolling and finishing of the rails, must contribute its part to secure the highest quality of the product incident to the chemical composition.

"7. Specifications should be drawn to indicate some of the major necessities of the consumer, and the tests and inspection conducted in a spirit to aid and invite the co-operation of the manufacturers to meet the progressive requirements in rail steel."

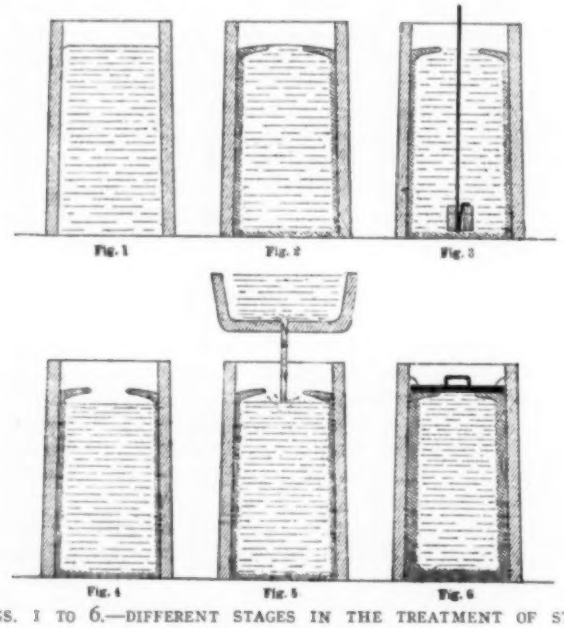
The Use of Anti-Piping Thermit In Casting Steel Ingots

The second paper of the session was presented by **Dr. E. A. Beck,** of the Goldschmidt Thermit Company, New York City. It deals with the interesting new application of anti-piping thermit for producing sound steel ingots. In contradistinction to an older method of using anti-piping thermit (in which a can filled with thermit was introduced into the upper part of the ingot and was intended to produce a heating effect), a new method has been more recently evolved in which the can with thermit is introduced as far down into the ingot as possible and simply a stirring action is intended proceeding from the

bottom to the top. This method has already been described in some detail in a paper by Dr. C. Canaris (this journal, Vol. X, page 232, April, 1912) and in a paper by Dr. Hans Goldschmidt (this journal, Vol. X, p. 739, Nov., 1912).

The application of the method is very clearly shown in Figs. 1 to 6, reproduced from Dr. Beck's paper and illustrating the different stages in the treatment of steel ingots with anti-piping thermit:

Fig. 1 represents a section of the mold filled in the usual way, either from the top or bottom. This mold



FIGS. 1 TO 6.—DIFFERENT STAGES IN THE TREATMENT OF STEEL INGOTS WITH ANTI-PIPING THERMIT

stands until a certain cooling off of the steel has taken place. The degree of the cooling can be determined approximately by watching the formation of the crust on the surface.

Fig. 2 shows how far the crust should extend before the introduction of thermit, in order to get the best results. In the case of a 5-ton ingot it takes about 7 minutes for the crust to form properly.

Fig. 3 shows the can introduced. The can is quickly pushed down through the steel to the bottom of the mold and kept there until the reaction is over. The resulting seething of the steel will last about 5 seconds.

Fig. 4 represents a cross-section through the mold after the reaction has taken place. The empty space which formed indicates that the steel has become denser.

Fig. 5 illustrates the addition of new steel. Care must be taken that the added liquid steel does not rise over the edge of the solidified crust.

Fig. 6 shows the treated ingot after the lid has been properly put on.

The advantages of this use of anti-piping thermit are due to I, improved quality, and II, better results in rolling.

I. *Quality.*—1. The steel becomes denser in proportion to amount of metal added after the thermit has been applied. The increased density is caused by the expulsion of the occluded gases due to the ebullition following the thermit reaction.

2. The removal of the already formed segregation toward the middle of the ingot and by their transfer to the surface of the ingot; further, segregations are avoided through equalization of the temperature of the steel.

II. *Rolling.*—1. Elimination of the laps which occur from piped ingots, or from ingots with a sunken or irregular head.

2. Better physical properties of the steel in the heads.

3. A saving of from 3 to 10 per cent of the finished material will be obtained. An average of 5 per cent would give a profit of about 75 cents per ton.

4. Elimination of defective plates caused by large blow holes on secondary pipes.

5. In small ingots the cost of the treatment amounts to about 35 cents per ton in the open-hearth department; against this there is an estimated saving of about 75 cents per ton in the rolling department. In large ingots (about 8 tons and more) the cost of the treatment is only 20 cents per ton and the saving is the same as in a small one.

6. A still further saving, which could not be calculated exactly, will be derived from the decreased percentage of defective plates, which is cut down to about 0.3 per cent, according to Dr. Canaris. In this case all plates which do not give the required dimensions or have not the desired physical properties in the head are classed as defective. Untreated ingots of the same group are often not uniform. Frequently there is a sound ingot, which will give good results without thermit treatment, and next to it there may be one with a large pipe or blow hole. Uniformity is secured by the thermit treatment.

Commercial Production of Sound Steel Ingots

A paper by Mr. **Emil Gathmann,** of Baltimore, described his process for producing sound steel in an economical and hence commercial manner, which is adaptable to the production of practically all steel manufacturers, by readily effected rational changes in the methods of casting, cooling, and subsequent handling of the ingots.

The process is based on the observation that the freezing or solidifying of an ingot which has been practically deoxidized or, as the term is used, "killed" in the mold, depends entirely upon the shape of the horizontal cross-section of the ingot at its various planes from top to bottom, and also upon the thickness and consequent heat-absorptive power of various parts of the mold walls.

This fact is clearly brought out and in a series of photographs in Mr. Gathmann's paper. As a result of these facts observed in practice, Mr. Gathmann designed a metallic mold constructed to accelerate the cooling of the lower or greater portion of the molten mass and teemed ingot (approximately from 80 to 85 per cent) and to retard the cooling of the uppermost portion of the ingot, thus causing the upper portion to remain liquid longer and to act as a feeder. The upper portion of the ingot does not actually remain liquid much longer than in the usual practice for similar cross-sectional area, but as the cooling of the lower portion is greatly hastened a differential in cooling is obtained, which is really what is to be desired.

According to Mr. Gathmann's method it is essential that there be considerable lag in the cooling of the upper portion of the ingot compared with the time of freezing of the lower portion, hence, the distance from the vertical central axis of ingot is made much less at the bottom of ingot and progressively increases toward the top of the ingot. In crucible steel ingots, which are largely made in split molds, the actual sectional area of the top and bottom may remain practically constant throughout the length of the ingot, but the distance from the vertical longitudinal axis to the surface of the ingot progressively increases toward the top. In ordinary big-end-up practice, where sufficient taper or differential in distance from the vertical axis to the surface of the ingot is given, to accomplish any notable reduction in depth of piping, the actual cross-sectional area of lower part of the ingot is much less than that at the upper part, hence the depth of pipe is not the true index of the actual volume or weight of cropping necessary to obtain physically sound steel.

One of the advantages of the system or type of ingot is that it is possible to obtain a practically uniform cross-sectional area at top and bottom of ingot and still obtain the benefits of the big-end-up type of mold.

Lifting of segregation is generally conceded to follow the

reduction of pipe, and where the pipe is lifted the steel below undoubtedly becomes more homeogenous and freer from segregation.

For open-hearth practice with the big-end-up ingot, one of the greatest difficulties has been to devise a method of stripping and handling the ingots.

Fig. 7 shows the Gathmann ingot mold and stool on a car or buggy after teeming. The big-end-up mold will

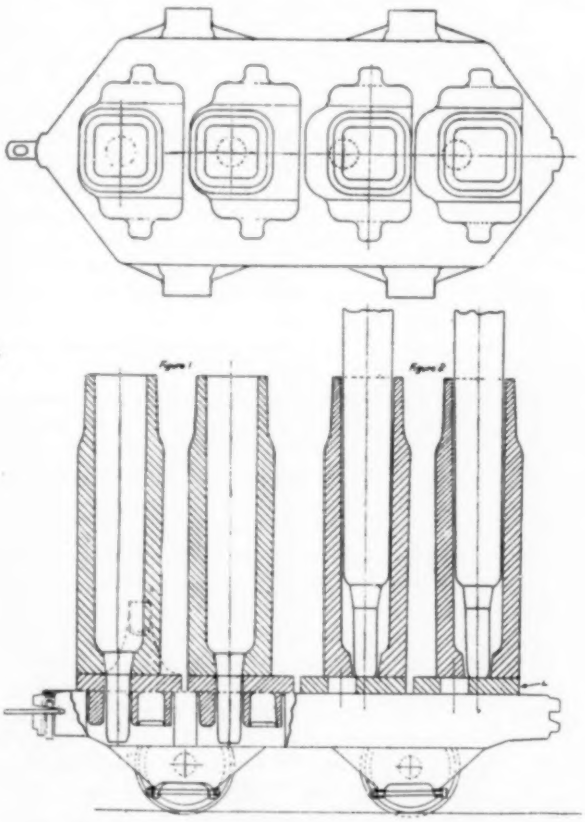


FIG. 7.—INGOT MOLDS FOR PRODUCING SOUND CASTINGS

require a special type of stool to avoid such difficulties as might ordinarily be experienced in stripping, due to fins forming at the base of ingot and locking it to the mold. A downwardly tapered plug, *c—c'*, seals the base of the mold and projects through the stool, *p—p'*. When the teeming is finished the metal, by reason of the wedge-like taper at the lower portion of the ingot, is forced into virtual contact with the walls of the mold, which, due to the thickness of same, rapidly absorb the heat of the ingot. A slight differential or reverse taper of the ingot cavity is made from 15 to 20 per cent from the top of the mold. The ingot in shrinking will automatically provide an air space at this portion, thus breaking the contact of ingot from mold walls and retarding the flow or loss of heat from this portion of ingot.

Other illustrations in the paper, not reproduced here for lack of space, show a modification of the Gathmann mold for bottom-cast ingots, a stripper with provisions to suit the stripping of the big-end-up mold, and a slab ingot mold for plate or sheet mills desiring to roll directly from the ingots.

The author concludes that the general practice employing this process "reduces the pipe in the deoxidized or killed steel so that with an average discard of 12 per cent sound metal will be obtained.

"Segregation will undoubtedly be disposed of in direct ratio to the lifting of the pipe." . . . "This system is not in an experimental stage as many tons of steel are daily being cast in both open-hearth and crucible plants in accordance therewith."

Production of Solid Steel Ingots

A paper by **Mr. Benjamin Talbot,** of Middlesbrough, England, with the above title was presented in abstract by Mr. Charles Kirchhoff.

It was essentially the same paper as presented by Mr. Talbot in a special meeting held on November 7, 1912, in New York, and reported in our December, 1912, issue (Vol. X, page 772).

To supplement the information given in that paper and reply to some criticisms made at the former meeting, Mr. Talbot had communicated the following remarks. In the discussion on his paper very great surprise was shown at the fact that in the sulphur prints exhibited there was a regular area of higher carbon and sulphur metal upon the inner wall of the solid envelope, and a purer center inside that.

The question was then raised whether this formation was regular when the ingots were treated uniformly, and doubts were expressed of its being consistently found. Further experiments have shown that it can be consistently relied upon, and there is no freak in the formation of this structure.

A photograph of the structure of an ingot is given in the paper, showing a gradual gradation of one structure to another. Results of drop tests and tensile strength tests are also given.

"Assuming that a sound rail with this structure (a hard working face, harder ring at back of this, and softer center) is accepted by rail users, we have then to consider the practical adoption of the process in steel works. There is no doubt that for open-hearth steel rails the large 250-ton tilting furnaces, giving heats of from 50 to 60 ton, irrespective of how the furnaces are operated, can give the steel in a more regular sequence, and are under better control as regards time of tapping than the so-called 60-ton fixed furnaces.

"As to the casting department, I may say that in my opinion the present ordinary methods have to be revised, and the casting must become an important department of steel manufacture. To-day both metallurgists and works' operators have neglected this department, and have not considered it as of anything like the same importance as the actual manufacture of the pig iron, or the conversion of the iron into steel.

"There is no doubt that the railway engineers have hit upon that point in the manufacture of steel rails which is causing most defects, and if given sound ingots the troubles now encountered in rail manufacture will be avoided. To my mind the casting department should have the attention of men skilled in the art to remove the slur now put upon steel makers in reference to cavities and blow holes, which form in the ingot during the solidification of the steel in the mold.

"I fully realize that if liquid compression of the ingot is to be adopted steel furnaces must not be allowed to 'bunch' together, and to tap a large number of heats at once as is the present practice. If a continuous set of rolls be used for liquid compression, then such rolls must be equal to dealing with the maximum quantity of steel tapped in any given time, but speaking as a work's engineer and practical operator, I see no difficulty in designing a plant to squeeze the required number of ingots to keep a modern rail mill supplied.

"Assuming that the necessary preparatory treatment has been made, we find that the size of the ingot must be standardized when such treatment is used and a 4-ton ingot seems to be a most useful size in this respect.

"Finally, in my opinion, the characteristic structure developed in rails treated by my squeezing process must cause these rails to be better than rails from the upper portion of ingots from ordinary heats which are always more or less segregated in the center."

Heating the Top

The Hadfield process for producing sound ingots by heating the top in a special manner was the subject of a paper by **Sir Robert Hadfield,** presented by Mr. Henry D. Hibbard.

Sir Robert Hadfield's process was described and illustrated in our November, 1912 issue (Vol. X, page 738), and his arrangements for applying the method on a large scale were described and illustrated in our February, 1913, issue (Vol. XI, page 79).

Mr. Hibbard gave an outline of the process, and exhibited a great many photographs and diagrams.

Comparative Notes on Steel Rail Rolling

A paper by Colonel **Robert W. Hunt,** of Chicago, Ill., was available in mimeographed form.

"I have frequently stated that while the chemical composition of steel was important, yet even greater importance was connected with the mechanical and heat treatment of the metal. During the past year I encountered such a positive example of that fact that I deem it worth putting upon record.

"A prominent railway system divided an order for open-hearth steel rails between two steel works, both of which are under the control of the same corporation, giving to one about 18,000 tons, and to the other 7500 tons of the same section and to be made under the same specifications. The rolling results obtained in two mills varied so widely that a study of the figures is intensely interesting and serves to illustrate the advantages obtained by careful ingot casting, and quite as pertinently the possible benefits of careful subsequent heating and rolling with moderate reductions in the rolling process. The smaller order was rolled complete in four installments during the same months that the larger order was being made in eight separate installments; but, to permit of exact comparisons, the totals of the four installments (completing the order with the mill which I will call A) are given with the totals of the first four installments of the larger order, made by the mill designated as B. These rolling results are:

	Mill A.	Mill B.
Number of rollings	4	4
Total number of rails rolled	18,278	27,832
Percentage of rails cut to short length because of flaws near ends, etc	0.7	7.6
Percentage of rails made second quality for flaws, etc	0.9	6.3
Percentage of rails scrapped for flaws, etc.	0.6	4.5
Percentage of rails scrapped for failure at drop test	none	3.4

"It should be stated that the figures above shown cover all the reasons for putting the rails in the classes stated. Thus a part of the percentage of rails cut to short lengths may have been because of bad drilling or bad sawing. Such classification, however, in this case is entirely proper, as neither mill suffered from unusual or abnormal difficulties in any way, and the figures indicate ordinary performance uninfluenced by unusual errors of either workmanship or mechanical troubles in rolling.

"Emphasis should be laid on the fact that both mills were working to exactly the same specification, and producing a section which has been in use for several years and in large tonnage. It is an 85-lb. one, having 36.7 per cent of metal in the head, 22.2 per cent in the web and 41.1 per cent in the base, being, therefore, well proportion for the avoiding of torn flanges or other rolling difficulties sometimes encountered with sections having thin flanges."

The chemical composition and drop test specified with average results obtained in the two mills are given in the paper. Both mills took advantage of the full range permitted in the chemical compositions. Though with respect to obtaining consistent carbon results, the results of mill A were more consistent than those of mill B, the "chemical composition obtained at the two mills agrees so closely that some other reason must be sought to explain the divergent physical results obtained in the product, and, therefore, data on the actual performance of the mill operations is important."

This is given in form of various tables in the paper.

It appears that "both mills were casting large heats, approximating 85 tons. These were made by almost identical methods of the usual scrap and pig-iron process, the iron taken from mixers, as required, and varying in proportion to the scrap used. As far as possible, at both mills, this mixer metal was used for recarbonizing in the furnace, but many heats had coke or coal added to the ladle on tapping, in addition to the usual ferro-manganese and ferro-silicon. While the heats at mill A were slightly larger than at B, the difference is not of importance, but it is pertinent to note that there are but six furnaces represented at mill A, as against 27 which furnished the rail steel at B; and, consequently, there were probably but two steel melters working at A, as against at least six at B.

"The personal equation may, therefore, have played an important part in making the steel.

"Equally noticeable is the fact that while the average for the time intervals of the various operations at the two mills is not much different, still the range between the maximum and minimum for B are consistently greater than for A.

"There can be but very little doubt that a delay between casting and stripping ingots and between stripping and charging in the soaking pits is likely to be seriously reflected in the soundness of the ingots, and of at least equal importance is the necessity for teeming the steel at uniform temperature. A variation of 50 minutes in the time heats were held in the ladle prior to casting must have caused variable teeming temperatures and produced many blow holes in some of the ingots. Admitting the presence of blow holes near the surface of the sides of the ingots, and remembering the oxidizing action in the soaking pits, it is not surprising that heavy reductions in the blooming mill had an extremely detrimental effect on the product.

"This is reflected, no doubt, in the large number of rails found containing flaws at mill B.

"Appreciating, therefore, the probable difference in the ingots produced at the two mills and the actual difference in the blooming practice, the principal other variable existing was in the reheating of the blooms at mill A. There the ingots were cut into four blooms, which were then given a wash heat in reheating furnaces after which they were rolled into two rails in a rail train of eleven passes. At mill B the rail train consisted of nine passes with no reheating of the blooms."

The reported history of the heats rejected at mill B is also given, together with analyses of the rejected heats.

Two heats were rejected at the drop test because two of the three test pieces broke on the first blow of the tup. The steel of the first heat was reported as being somewhat low in temperature when tapped. While pouring, the nozzle froze up and while the ingots were in the soaking pits an average of four hours and twenty minutes, they bloomed cold and rough. The second heat was quite the opposite, the tapping temperature being high and while casting the stopper head was finally lost, but the heating and the blooming were normal.

Four heats were lost at the drop test, because they exceeded the deflection limits. On the first, all conditions were reported normal, save that the ingots were held in the pits for an average time of 24 hours. The second heat tapped cold, the pouring nozzle froze, and the ingot tops were spongy. The third tapped hot, but otherwise, like the fourth, had normal conditions.

General Discussion on Sound Ingots

A general and quite extended discussion on improving the soundness of ingots followed.

The following four questions on piping had been formulated by the committee of arrangements as a basis for the discussion:

1. Is the present method of getting rid of the pipe by cropping a safe and reliable device for making sound steel?
2. Is it desirable to increase the amount of metal arbitrarily cropped off the top of an ingot, and if so what would be a fair excess price to pay for cropping off 20 per cent?
3. Is it commercially practicable to make ingots without pipes or blow-holes and what addition of expense of manufacture would be justified to accomplish this result?
4. What process for making pipeless ingots seems the most promising of commercial success and why?

Dr. **A. Sauveur,** of Harvard University, opened the general discussion by referring to the fact that years ago he had pointed out how great a source of danger the pipes in ingots were for rails, but his word of warning had met only with opposition and it had been said that the pipes would be welded in manufacture of the rails, etc. It was a great advance that now both rail makers and railroads were willing to admit the danger. If the processes for producing sound ingots, described in the preceding papers, are right or if any one of them is right, there will be no more excuse in future.

Mr. **Robert Job** commented on the questions which were the basis of discussion. He pointed out that with good mill practice very little piping is present, while with poor mill practce 50 per cent discard might not produce sound rails. For this reason the question (2) where it is desirable to increase the discard of metal arbitrarily should be answered without hesitation in the negative. Such practice would simply penalize good mill practice. Tests should be made to detect piping in any ingot to be used for rail steel. The tests of the Lehigh Valley Railroad were described. A confirmative answer was given by Mr. Job to question (3) whether it is commercially practicable to make ingots without pipes or blow-holes. He pointed out the need of thoroughly deoxidizing the steel; soundness presupposes deoxidation.

Mr. **Max H. Wickhorst,** engineer of tests of the Rail Committee of the American Railway Engineering Association divided rail failures into head failures (split heads,) base failures, and "broken rails" (square or angular breaks). The first class—that of head failures—makes up one-half of all the rail failures and is due to the interior condition of the ingots from which the rail was made. Segregation is the source of the trouble. Segregation must be distinguished very sharply from piping. Silicon, titanium, aluminium reduce segregation. When they are used, the ingot shows a deep pipe, but less spongy condition. The origin of split heads of rails is not piping, but the condition of segregation. We need not be so much afraid of the pipe itself, but rather of segregation. If this is avoided 50 per cent of all the rail trouble is solved.

Mr. **L. E. Howard,** of the Simonds Manufacturing Company, described a method of his own used successfully by his company in crucible steel practice. The process employs essentially lateral compression. Two sample ingots showing the successful suppression of piping were exhibited. In reply to question three he pointed out that his method was a commercial success, as the average cost including everything is less than 3 per cent of the cost of the ingots. For ingots of larger size his method must be somewhat modified.

Mr. **J. E. Sague,** engineer of the Public Service Commission of the Second District, summarized the chief results of the report which he had made as a result of the inquiry into the limits of the rail failure proposition. He found that from the accident standpoint the failure of rails is very much exaggerated in newspaper discussions. Nevertheless, as it really is, it is a grave proposition and a great deal more knowledge is necessary on the subject. As far as the steel maker is concerned, the start must be made at the ingot. One cannot expect sound rails, except when starting with sound ingots. But the railroads have also their own problems to solve. It is a great advance that the fact has been recognized that a rush from New York to Chicago in 18 hours is not safe under any and all conditions of weather. To have their trains arrive on time, not to rush them through in the shortest possible time, should be the ideal of railway managers. In the whole there is no reason for despair.

Mr. **Henry D. Hibbard,** of Plainfield, N. J., commented at some length on the Hadfield process and then made some remarks on segregation. The problem is probably more serious than usually imagined. The segregated material heretofore used for analysis is probably always diluted by admixtures.

Mr. **James E. Howard** pointed out that in the first few passes there is evidence of elimination of the possibility of pipes and blow-holes. But what cannot be eliminated at all are particles of slag, etc., disseminated through the steel. Against them there is no correction. It is most important that these minute slag inclusions should be avoided.

Mr. **W. C. Cushing,** of the Pennsylvania R. R. Co., said that question (2), whether it would be desirable to increase the amount of metal arbitrarily cropped off the top of an ingot, should surely be answered in the negative. Progressive discard upon examination of soundness of the ingots seems to be the best at present. As to the statistics of rail failures in the United States head failures are 60 to 70 per cent of the total failures, web failures 5 per cent, base failures 5 to 10 per cent, broken rails 20 to 25 per cent. The rail failure statistics of the United States includes all these failures, not simply broken rails, as European critics have wrongly assumed. It is the broken rail that gives the most trouble and thought. Mr. Cushing asked finally as to the relative efficiency of aluminium, silicon and titanium as deoxidizers.

Mr. **G. H. Clamer,** of the Ajax Metal Company, described an electric device based on the principle that if the top of the ingot is kept hot, no piping will take place. There are different methods possible for keeping the top hot. The device described by him makes use of the pinch effect (as in the Hering electric furnace). It consists chiefly of a magnesia block with a cavity and a narrow channel connecting this cavity with the outside. This block is placed into the metal at the top of the ingot so that the steel enters the block through the channel. By providing suitable steel electrodes with connections to a small transformer an electric current is passed through the metal, the pinch effect is produced in the channel, and a heating and circulating effect is obtained as in the Hering steel furnace.

Dr. **A. S. Cushman,** of Washington, D. C., spoke on the extended investigation which is to be carried out under his direction by the American Rolling Mill Company. No final results have yet been obtained, but a special ingot splitting apparatus has just been installed and with this facility it is hoped to get decisive results as to the effectiveness of the different methods.

Mr. **N. Petinot,** of the Titanium Alloys Manufacturing Company, first dealt on the significance of some of the segregation analyses mentioned in Dr. Dudley's paper. He then discussed the relative efficiency of aluminium, silicon and titanium as deoxidizers. While all three remove oxygen, the results of the deoxidizing reactions are different. Aluminium is oxidized to alumina and the alumina particles are liable to remain disseminated in the steel and cause trouble. Silicon forms silica, silicides, silicates. With titanium on the other hand titanic acid is formed which rises to the top and is eliminated. Silicon should be used moderately and should be followed by titanium which will not only complete the deoxidation but will remove the iron silicates. Dr. Dudley, later in the discussion, confirmed the efficiency of titanium as a deoxidizer.

Prof. **Henry M. Howe** discussed the reason why pipes are formed at all and emphasized that it was all important to let solidification of the top lag behind the solidification of the bottom. Millmen have not gone to the limit in this respect. To cast from the bottom is worst in this respect, to cast from the top with the large end up is the best. There are special methods of heating the top, but they represent necessarily some expense. But the same result may be obtained by purely administrative methods. As to pouring, it may be asked whether it is possible in pouring to fill first only part of the mold and then come back a little later and pour again and fill up the top. As to the possibility of this, millmen may say no; but if yes as answer is at all possible, it would be well worth while trying it. In the whole purely administrative methods may perhaps prove just as good as special and more expensive processes.

Mr. **K. W. Zimmerschied,** of the General Motors Company, divided the troubles which remain in the steel from the ingot, into pipes, segregation, and enclosures of non-metallic material. Pipes cause the smallest trouble, the enclosures of non-metallic material (oxides of iron and manganese, alumina, etc.) cause the most trouble. To avoid them is a more important problem than that of piping. If aluminium is used as a deoxidizer properly, the alumina, though solid, will probably rise as quickly to the top as titanium oxide on account of its much lighter weight.

Mr. Zimmerschied then took up the four questions which formed the basis of discussion. Question (1), whether the present method of getting rid of the pipe by cropping is a safe and reliable device for making sound steel, he answered with yes. In reply to question (2), whether it was desirable to increase the amount of discard arbitrarily, he said that this was out of the question. Question (3), whether it is commercially practicable to make ingots without pipes and blowholes, he answered with yes and added that the amount of the additional cost was a commercial consideration depending on the circumstances. As to question (4), what process for making pipeless ingots seemed the most promising of commercial success, he said that the cure must be a continuous one and must progress from the bottom to the top. In conclusion he made what he called a "wild suggestion": with our advance in cutting methods, why not split every ingot before you roll it?

Mr. **Henry Hess,** of the Hess-Bright Manufacturing Company, described some experiments made by himself with a method used formerly in a similar way by Professors Howe and Stoughton. To look into the interior of an ingot while in course of solidification, he made solidification experiments with resin in test tubes. He could form pipes of any form and at any place by changing the method of heating the test tube. Where you heat it, and where you heat it last, there you get the pipe.

After conclusion of the discussion lunch was served in an adjoining room.

Wednesday Afternoon Session

Mr. John Birkinbine presided during the afternoon session.

A paper by Mr. **James R. Finley,** of New York City, on the "valuation of iron mines" was read, in the absence of the author, by Mr. Stoughton and discussed by Messrs. Birkinbine, Norris, and Parker.

Dr. **Albert Sauveur,** of Harvard University, then presented a paper entitled "Notes on Cast Iron," with special reference to the recent paper of Mr. J. E. Johnson, Jr. We reserve an abstract of Dr. Sauveur's paper for a future issue. The paper was discussed by Dr. Howe and Dr. Moldenke.

Dr. **Henry M. Howe,** of Columbia University, then presented his paper, "Why does lag increase with the temperature from which cooling starts?" We reserve an abstract for a future issue.

Sintering Iron Bearing Materials

A paper by Mr. **B. G. Klugh,** of Birdsboro, Pa., on "The microstructure of sintered iron-bearing materials" was read, in the absence of the author, by Mr. Arthur S. Dwight and illustrated by a long series of lantern slides of structural photographs. The chief results of the paper are as follows:

The permeability of the cell-wall of a sintered product varies, inversely, as the degree of fusion to which it has been subjected.

In a product of complete fusion, the silica present combines with its equivalent of iron oxide to form a perfect glass which, from its greater fluidity, envelops and seals up the remaining iron oxide from the action of gases.

Conversely of the foregoing conclusion, in the product of the lowest degree of fusion, the iron oxide and slag-forming materials as a unit are bonded together by incipient fusion, leaving the predominant iron oxide, free and vulnerable to the action of the gases, in the highest degree attainable in solid products.

The above salient facts show that the Dwight and Lloyd products when properly made, do possess those properties which distinguish it from the products of other sintering processes or agglomerating methods, by freedom from those constituents, to which scouring action in the blast furnace is attributed.

New Design of Open-Hearth Steel Furnace Using Producer Gas

Mr. **Herbert F. Miller,** Jr., of Verona, Pa., in his paper on this subject, expressed the opinion that the gas and brick costs of open-hearth furnaces using producer gas could be greatly decreased by a change in the design of the port, which would materially reduce the first cost of the furnace, the rebuilding cost, and the repair cost.

He first outlined the defect of the present design and then described two new designs, both of which involve the same principle of putting a single air uptake directly back of the gas uptake as is shown in Figs. 8 and 9.

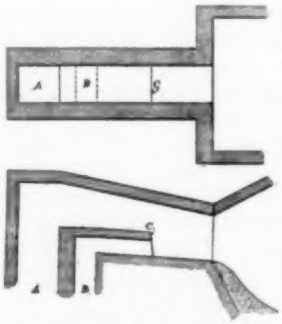


FIG. 8.—PART OF O. H. FURNACE USING PRODUCER GAS

In Fig. 8 the port has about one-third of the width of the hearth, whereas the present type has the full width. The roof of the port dips sharply, compressing the air down on the gas; and the roof of the hearth is gradually dropped to meet it. The gas-port arch is retained although flattened to the width of the port. The end C is from 6 to 10 ft. from the edge of the bath.

There is a small seal-door on each side of the port at C to permit the repair of the port or the removal of the brick which might fall from the gas-port arch.

The point C will move back slower than in the present type, because it will be less exposed to the flame, which will always be under the air.

The possible defect of this design is that when the gas-port arch has become short, combustion will commence too far from the bath, and make a high gas cost. Another objection is, that this design could not be applied to existing furnaces because their ports would have to be lengthened considerably. If the flame in the above design can always be controlled, even when the gas-port arch has burned back to the gas uptake, why have a gas-port arch at all?

The design shown in Fig. 9 is the solution. Here it will be seen that the furnace is similar in outline to the present open-port type in use at Homestead. The air uptake A is back of the gas uptake, from which it is separated by a substantial wall. The gas uptake, which is as wide as the port, is from 6 to 10 ft. from the bath. The port is open and smooth, narrow and low. The small cross section of this port assures a speedy union of the preheated air and gas, resulting in a short flame of intenst heat, which will be under control throughout the run.

Some may think that the gas rising at right angles to the air will make a high flame. If the port is low enough this will not occur. The author has put a stream of air having a pressure of 70 lb. per sq. in. under a natural gas flame, and found that there was very little deflection of the flame.

This design could be applied to the present type of producer-gas furnaces. The only alteration of importance would be a reversal of the gas checkers from the outside to the inside position.

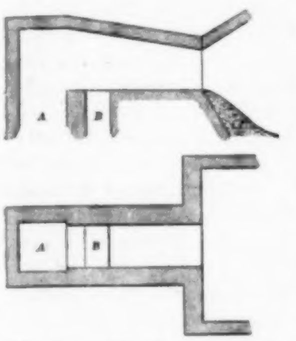


FIG. 9.—PART OF O. H. FURNACE USING PRODUCER GAS

In building a new furnace, a great saving of brick would be made in the ports, as can be readily seen.

Repairs to the dividing wall can be made through small seals by the side of the gas port. The flame will always be under control and the furnace should slow up only because of dirty checkers and insufficient draft. Natural gas could be used if desired by introducing it as usual at the sides and in front of the uptake *G*. In the author's judgment, the adoption of this design would do away with much of the trouble and expense attending the present type of producer-gas furnace and greatly decrease the operating cost.

In the discussion Mr. Hibbard questioned whether with this design a good spreading of the flame over the hearth could be obtained. Mr. Miller replied that the design worked well with natural gas.

The last paper on the program was presented by Mr. **F. L. Grammer,** of Leesburg, Va., on "Blast furnace slag analyses for 24 hours."

The meeting then adjourned.

Notes on Chemistry and Metallurgy in Great Britain

Institution of Mining and Metallurgy

At the third general meeting of this Institution, held at the rooms of the Geological Socity, Burlington House, on Dec. 19, three papers of unusual interest were discussed together, as all bearing upon questions connected with tin metallurgy. These were Mr. **W. Fischer Wilkinson's** "On the Dressing of Tin Ores in Cornwall," Mr. **H. J. B. Rawlin's** "Notes on the Direct Volumetric Determination of Tin," and Mr. **R. T. Hancock's** "Notes of the Valuation of Nigerian Tin Concentrates."

Mr. Holloway, who opened the discussion, pointed out that as Mr. Rawlin's paper dealt entirely with the analysis of metallic tin, it might at first sight appear to be more a paper for an analytical society than for themselves; but that it was important in connection with the chemical testing of the metallic tin obtained by such methods as the old Cornish, or culm method, or the more modern cyanide method of determining tin in black tin concentrates. Mr. Wilkinson's paper related entirely to Cornish tin dressing. Mr. Holloway was sorry to see heavy Californian stamps so largely used in Cornwall. The recovery of the tin was rendered less direct by the use of heavy stamps.

The discussion generally treated of the most correct method of estimating the recovery, and the applications of the chemical assay and the "vanning" test. Mr. E. S. King said that he found the only way to get at the correct extraction was by the chemical assay of the heads and the output of pure black tin. The tin account was kept in just the same way as a gold account; on one side was entered so many tons of ore containing so many tons of black tin; against that they had the amount of pure black tin sold. This was at Carn Brea.

With reference to the vanning assay, he said the most obvious explanation of tin which escaped from the vanning and was caught by the wet assay was the smallness of grain of cassiterite in the tinstone. It should be remembered, also, that there were tin-carrying minerals other than cassiterite.

Mr. Josiah Paull estimated the recovery at the mine with which he was connected at 78 per cent to 80 per cent. Mr. Wilkinson had given the average for Cornwall as something like 50 per cent to 55 per cent. His results were from samples of pulp from the screens taken practically every hour throughout the twenty-four. Afterwards replying to Mr. Flowers, Mr. Paull modified his percentage by a small allowance for wet weight.

Mr. H. L. Twite, in a written communcation, pointed out the importance of even 1 per cent in recovery. He had known samples sent to two different assayers and the difference in the assays returned was 0.3 per cent, which was repeated on several assays.

Mr. Amos Treloar gave the recovery for the Carn Brea and Tincroft plant, one of the oldest and most out of date in Cornwall, as 73.7 per cent. Mr. Frank Merricks inquired if Mr. Paull, in his figure of 78 per cent to 80 per cent of recovery at South Crofty, meant a 62 per cent extraction?

The irregularity of results quoted and the disagreement between the figures obtained on assay and test show how extremely difficult it is to obtain a standard method of sampling and testing which would admit of exact comparison.

Oil Fuel

An example of the growing interest in oil fuel in British scientific circles is afforded by the lectures on the subject delivered at the Society of Arts by Prof. **Vivian B. Lewes** (Cantor Lectures). On the question of fuel oil for power purposes, the professor noted that the method of obtaining the heat from the fuel had evolved into spraying the oil into the furnace, with a swiveling motion to the cloud formed, thus retarding the passage through the furnace and combustion space, so as to give sufficient time for complete combustion. When used as a fuel for raising steam, petroleum might be taken as having an evaporative power, weight for weight, 50 per cent higher than Welsh steam coal. It was known that 9.5 lb. of water evaporated was the best duty which could be obtained per pound of coal, which is a difference of 4.5 lb. from the calculated amount, due to losses of heat. Allowing the same ratio of loss for the oil fuel, the evaporative duty should be 13.2 lb. in practice, or even higher, according to the type of boiler and length of time under steam.

With regard to the internal-combustion engines, taking the highly efficient Diesel engine as an example, in spite of its only consuming one-third of the weight of fuel used by the best over-type superheated condensing steam plant, yet for a given power with coal at 18/- per ton and oil at 42/- per ton, the annual cost of running the steam plant would be less than that of the Diesel engine when all items of expense were taken into consideration. On surface combustion, Prof. W. A. Bone had found that when a mixture of a combustible gas with air was forced under pressure through a porous diaphragm of asbestos and fireclay, if the mixture were ignited as it leaked through the porous mass, it could not explode, but burnt on the surface of the diaphragm and quickly raised it to a very high degree of incandescence. This had been adapted for practical work with a boiler, and a very high efficiency had been obtained.

Steel Metallurgy

"Recent Advances in Steel Metallurgy" was the subject of a lecture delivered at the Royal Institution by Prof. **J. O. Arnold,** of Sheffield, in January. After noting the improvements in steel for cutting tools introduced by Mushet, and other modifications of the same process, involving the use of tungsten, the lecturer goes on to the introduction of vanadium steels in 1899-1902. The addition of about 0.6 per cent of vanadium to a pure carbon steel containing about 1 per cent of carbon raised the yield point from 35 to 65 tons per square inch, and the maximum stress from 60 to 86 tons, the elongation being 7 per cent, and the reduction of area 8 per cent.

According to researches made by the lecturer and Prof. A. A. Read, of the University of Wales, vanadium did not seem to form a double carbide with iron. It gradually wrested the carbon from the carbide of iron till when about 5 per cent of vanadium was present Fe_3C could not exist, and only a vanadium carbide V_4C_3 containing 15 per cent of carbon was present, this constituent being constant at any rate in tool steels containing up to 14 per cent of vanadium.

Strains in Metals

Attention is called by Mr. **T. Vaughan Hughes,** A. R. S. M., in the *Electrical Review* to strains set up in the manufacture of copper and other metallic parts, such as condenser tubes. He advocates that at least two causes of failure of metals and alloys under the control of the makers should be removed, namely (a) uneven heating appliances and (b) antiquated heat treatment in chemically and physically active atmospheres.

Oil Explosions in Transformers and Switches

Two remarkable accidents, accompanied by fatal consequences, recently occurred at a main transformer station of the Victoria Falls & Transvaal Power Company. The station takes current at 40,000 volts from the trunk lines and transforms down to 20,000 volts with oil-filled transformers, the oil being cooled by constant circulation of cold water through pipes. "Expansion tanks" are attached to and above the transformers to ensure their being always filled with oil and to receive any outflow resulting from increase of volume of the oil when its temperature rises.

While a thunderstorm was in progress there was a "surge" on the main which broke down one transformer with the result that it was cut out. Three attendants examined the transformer and one of them, wishing to see the level of the oil in the tank, held a lighted match to the aperture at the top, when a violent explosion occurred which killed one man and severely injured the two others. It appears, too, that instantly after the explosion a large quantity of oil was ejected from the tank and set the transformer in a terrific mass of flames.

About a week later another explosion took place at the same station in a 20,000-volt oil switch controlling some low-voltage apparatus. The cause is attributed to a "surge" on the 20,000-volt lines which discharged across the busbars and made the oil switch trip; and the instantaneous effect was that a strong fireproof dividing door was burst open, a man at a telephone in the adjacent room was enveloped by the vast mass of flame and died soon afterward and others were seriously burned.

The commonly accepted explanation of the first of these explosions is that arcing, resulting from a "short" in the transformer, decomposed the oil with liberation of hydrogen or gaseous hydrocarbons, such as methane, or perhaps a mixture of these gases, which, rising into the expansion tank, formed an explosive mixture with the air therein. A similar explanation has also been offered for switch explosions. But in both cases the intense and rapid combustion of the ejected oil—which would, of course, originally have a high flashpoint, probably not lower than 160 deg. C.—suggests that there may also have been some action analogous to "cracking" which split up the higher liquid paraffins into paraffins of considerably lower density and flashpoint together with some olefins.

A chemical examination of the residual oil might throw some light on the question, which manifestly deserves more thorough investigation than it has yet received.

Detection and Estimation of Carbon Monoxide

Two methods of determining small percentages of carbon monoxide in air have recently been communicated to the Académie des Sciences of Paris. MM. **A. Lévey** and **A. Pécoul** adopted a suggestion of M. Armand Gautier and have found that the reduction of iodic acid by carbon monoxide between 60 deg. and 80 deg. C. affords a means of detecting one part of the gas in 200,000 parts of air and yields accurate quantitative results. Three liters of air are passed from the aspirator into a vessel containing iodic acid and maintained at a temperature of about 70 deg. C.; the iodine liberated is absorbed by chloroform, and its amount is determined colorimetrically against solutions of iodine, of known strength, in chloroform. The authors have successfully applied this process to the examination of air in Paris schools, hospitals and railways.

The other method, described by M. **Guasco,** is an application of the well-known fact that platinum black absorbs combustible gases, and especially carbon monoxide, with evolution of heat. It differs from earlier proposals to utilize this phenomenon for the estimation of carbon monoxide in the employment of a Leslie differential thermometer, one bulb of which is coated with platinum black, while both bulbs are protected by porous envelopes preventing direct access of air, but permitting osmotic penetration.

It is obvious that M. Guasco's apparatus can only be relied on when the air is ascertained to be free from combustible gases other than carbon monoxide. An apparatus on the same principle was devised by Messrs. J. Pitkin and T. Niblett in 1889; but they used two ordinary sensitive thermometers, side by side, one having the bulb coated with platinum black, and the instrument was not sufficiently sensitive for the estimation of less than about 1/10 per cent of carbon monoxide in air.

A New Method of Manufacturing Gold Leaf

A London firm has introduced a novel, patented, process of making gold leaf even thinner than the best leaf produced in the usual way, although the new leaf really consists of three metallic films. A wheel of aluminium, some 5 ft. in diameter and 5½ in. wide, has its periphery highly polished and then evenly coated with a soluble gummy material which, when nearly dry, receives a coating of very fine metallic powder. This metallic film is then polished and is electrolytically coated with a film of nickel by rotating the wheel with its lowest point making contact with the surface of a suitable solution of a salt of nickel containing the anode, while the rim of the wheel is the cathode. After washing, and again polishing if necessary, a film of gold is in the same way deposited on the nickel.

The composite film is removed by making one cut across its width, immersing the rim at this point in a liquid solvent of the adhesive substance and slowly rotating the wheel. As the film falls away a paper band, the motion of which synchronizes with the peripheral speed of the wheel, carries it off to be cut into slips and made up into books.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the year 1912 show considerable increases in both imports and exports as compared with 1911, except imports of new ships. Iron and steel, including manufactures, were imported to the value of £12,970,862, an increase of £1,837,008; and were exported to the value of £48,628,918, an increase of £4,898,626. Imports in other metals, including manufactures, amounted to £31,199,898, an increase of £3,618,654; and exports reached £12,299,149, an increase of £1,276,613.

In electrical goods, the imports were £1,457,646, a rise of £22,154; and exports went to £6,369,877, or £1,550,503 more than for the previous year.

Imports of machinery are put at £6,820,744, an increase of £1,052,082; and the exports at £33,161,772, an increase of £2,201,194. The value of imports of new ships was only £33,654 showing a decrease of £30,830; but exports touched £7,031,890, an improvement of £1,368,784.

Taking the figures for December alone, imports of iron and steel increased £248,235 and exports rose £628,433; imports and exports of other metals went up £313,718 and £154,136 respectively; imports of electrical goods declined £1,117, but exports rose by £102,900; imports of machinery increased £160,901 and exports were £92,431 higher; new ships showed a drop of £509 in imports, but exports improved by £20,591.

Market Prices—January, 1913

Copper opened at £76-13 and declined slowly till the 8th (76) when it dropped sharply, reaching £70 on the 14th, and after short recovery to £71, £68 on the 20th. Improved to £71 on the 23rd, but again dropped, selling £69-10 on the 25th and closing £68-15.

Tin opened at £229-10 and showed only slight falling off to the 8th, then became more jerky and touched £227-15 on the 14th, and again declined sharply on the 18th, reaching £226 on the 21st. Has since risen, closing £230.

Cleveland opened 67/5, rose by the 7th to 68/- and has since been weak, being under 65/- on the 17th and down to 65/3 on the 24th. Since unsteady, but keeping near this price, it closes 65/3.

Hæmatite opened 82/- and remained firm till the 17th when it dropped away to 80/5, since recovering slightly to 81/-; closes 81/3.

Scotch Pig has been in sympathy with Cleveland, opening 74/- and dropping to 70/10 by the 17th. It closes a little better at 71/3.

Lead (English) opened £18.7.6 and has shown unusual variations. It was £18.18 on the 8th, and declined thence to £17 on the 12th, recovering to £17.10 by the 23rd, but afterwards weak; closing £16.17.6.

Aluminium	£89. 0.0
Alum, lump, loose, ton	5.15.0
Antimony, black, sulphide, powder, ton	21. 0.0
Borax, British, refined crystal	17. 0.0
Copper ore, 10 to 25 per cent per unit	12/10½ to 13.4½
Copper sulphate, ton	25.10.0
Carbolic acid, liquid, 97-99 per cent, per gal	1.9
Creosote, ordinary good liquid, per gal	0. 0.3
Caustic soda, ash, 48 per cent, ordinary, ton	5.10.0
Hydrochloric acid, cwt	5.0
India rubber, Para	4.4
Naphtha, solvent, 90 per cent @ 160 deg. C., per gal	1.1
Petroleum, Russian, spot, per gal	8
Quicksilver, bottle	7.15.0
Sal ammoniac, lump, firsts, delivered U. K	44. 0.0
Sulphate of ammonia, f.o.b. Liverpool	14. 7.6
Sulphur, recovered, ton	5. 5.0
Shellac, cwt	3.12.6
Platinum, ounce	9. 5.0
Zinc, V. M., f.o.b. Antwerp	29.15.0
Tin, ore, 70 per cent, ton	£148 to 150. 0.0

Differences

Higher.		*Lower.*	
Aluminium	£1. 0.0	Copper ore	£0. 0.7½
Carbolic acid	4	India rubber	2¾
Quicksilver	6.6	Naphtha	1
Sulphate of ammonia	8.9	Shellac	11.6
Tin	1.10.0	Zinc	10.0
		Copper	7.5
		Cleveland iron	2.6
		Scotch pig	2.6
		Hæmatite	1.0
		Lead	1.12.6

Recent Chemical and Metallurgical Patents

Gold and Silver

Sand-Slime Classifier.—The separate treatment of sand and slime products in cyanidation has created a demand for efficient means of separating these materials. A recently patented device for this purpose is shown in Figs. 1 and 2, being the invention of MR. PHILIP ARGALL, of Denver, Colo. The classifier consists of an inclined trough E, in which are placed two spiral conveyors I which rotate upwardly and inwardly. The trough may be divided into three portions, namely, B, in which settlement of sand occurs, C where the sand is unwatered, and M, a filtering portion for withdrawing additional moisture from the sand. The inclination of the trough will vary from ten to twenty-five degrees, but for ordinary silicious ores probably will be about fifteen degrees. A weir G consists of a level strip over which water and slime may flow into a launder g. A feed-box is shown at A, provided with openings a', through which the mixture of sand and slime is introduced into the trough.

In operation the pulp is fed through the box A. The sand settles and is worked upwardly by the spiral conveyors, while the slime remains in suspension, aided by the action of the conveyers, and overflows at G. The inventor states that he has devised the weir so that the overflow from a feed of six tons per hour will all pass a 150-mesh screen. The action of the conveyers results in building a bed of sand, as shown in Fig. 2, and the slime and water draining from the sand bed finds its way into the side launders K' and runs back to the settling portion of the trough. The filtering portion consists

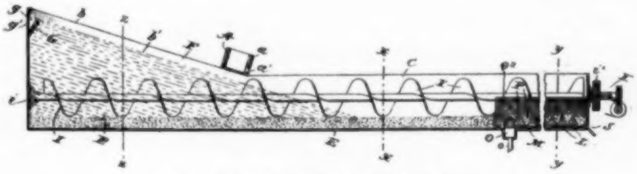


FIG. 1.—SAND SLIME CLASSIFIER

of angular slats L arranged transversely of the trough. If desired, a spray of water may be made to play on the sand as it passes over the filtering portion. In a modified form of classifier, the settling portion is circular, thus giving a larger weir. (1,044,844-5, Nov. 19, 1912.)

Slime Filter.—Various applications of a porous mineral septum in agitating and filtering mineral slime have been proposed by the advocates of the Just process, silica sponge being the material used. In patents recently granted to MR. W. W. ROBACHER, of Rochester, N. Y., assigned to the Just Process Co., Syracuse, N. Y., it is proposed to use silica sponge brick as a facing on a revolving vacuum filter, as shown in Figs. 3 and 4 (p. 160). The device consists of a drum 2 revolving in a tank 11 containing the slime to be filtered. The drum is covered on its sides with a porous mineral septum 6. The filter revolves on a hollow shaft actuated by gearing 12, 13. Vacuum is applied to the inner portion of the drum through pipe c, and causes a cake of slime to form on the faces of the drum, while cyanide solution is drawn through the septum and removed for precipitation. As the drum revolves and the cake is removed from the tank, it is removed from the filtering surface by means of scrapers 15 and transferred to a mixer 17, in which it is agitated with more solution or water and sent to a second similar filter. (1,051,160-1, Jan. 21, 1913.)

Copper, Lead and Zinc

Dry Chlorination of Mixed Sulphide Ores.—Among the recent efforts made to evolve a suitable process for treating complex sulphide ores, is one disclosed in a patent granted to MR. JOHN L. MALM, of Denver, Colo. The principal steps

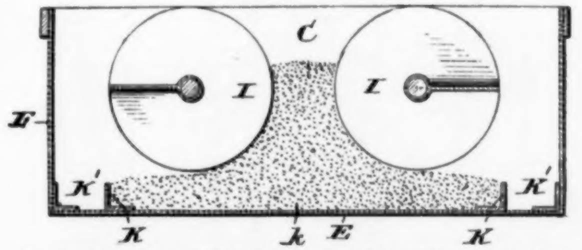


FIG. 2.—CROSS-SECTIONAL VIEW OF SAND-SLIME CLASSIFIER

outlined in the present patent are the following: Finely crushed ore is fed into one end of a tube mill where it comes in contact with dry chlorine gas admitted into the tube from the opposite end. The reaction between the sulphides and chlorine evolves heat, but the temperature is controlled either by varying the quantities of ore and chlorine admitted, or by admitting the necessary quantity of air from the outside. In either case the temperature is maintained below the point at which sulphur monochloride or plastic sulphur would be formed as products of the reaction, thus preventing the balling or caking of the ore, and permitting every particle of it to be

acted upon by the chlorine. This step in the process affects only a partial chloridization of the ore, so that some of the metals exist as chlorides in the lower state of oxidation.

The ore is then transferred to an agitator, where it is treated with water or mill solution containing free chlorine, whereby the chloridization of the metals is completed and all exist as chlorides in the high state of oxidation. These chlorides are soluble and pass into solution, while the gangue and sulphur remain in a solid, granular state. The mixture is then filtered, and the metals precipitated from solution in any suitable manner.

It may be added that the procedure disclosed in this patent does not represent the finished state of Mr. Malm's process, other features being contained in pending applications. (1,049,-746, Jan. 7, 1913.)

Hydrometallurgical Process for Copper Ores.—A process patented by Mr. Joseph Irving, of Salt Lake City, Utah, has for its object the treatment of copper ores containing precious metals. It consists essentially of the following steps: Treating a low-grade carbonate ore of copper with dilute sulphuric acid solution, injecting steam and mechanically agitating the mixture, drawing off the resulting solution, passing it through a filter of sand and pyrites, and precipitating it on iron. The precipitated solution, containing at this time ferrous sulphate, is oxidized by agitation with air and injecton of steam, and used for the treatment of a sulphide ore of copper containing the precious metals. Sodium chloride is added also, and the mass agitated mechanically and with steam. The resulting solution, containing copper, gold and silver, is then filtered and passed over iron and zinc to precipitate the contained copper and precious metals. On reoxidizing the precipitated solution and adding the requisite quantity of sulphuric acid, it is ready to be used on a fresh lot of ore with addition of salt. (1,048,-541, Dec. 31, 1912.)

Pyrometallurgy of Zinc-Lead Sulphides.—A method for treating mixed zinc-lead sulphide ores has been patented by Mr. C. A. L. W. Witter, of Hamburg, Germany. The ore is first finely ground and roasted until it is practically free from sulphur. It is then briquetted with a reducing agent and flux, as coal and lime, and fed into one end of a reverberatory furnace. The furnace is fired with gas, and in such a manner that the feed end contains a neutral or preferably re-

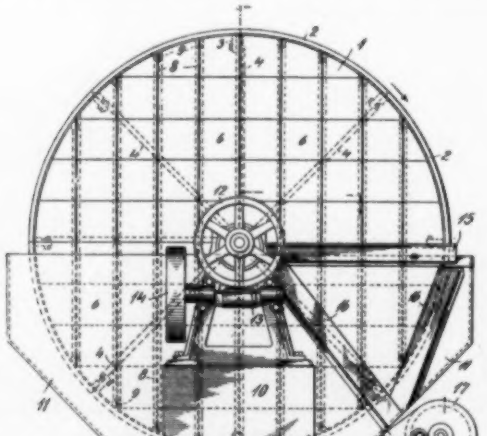


FIG. 3.—VACUUM SLIME FILTER

ducing atmosphere, while the opposite end is strongly oxidizing. The furnace contains a bath of slag on which the briquets fall. The metals in the briquets are reduced, the lead sinking through the slag bath, while the zinc vapor passes toward the opposite end of the furnace to the outlet. Here it is oxidized and subsequently caught as zinc oxide in flues and bags. The lead and slag may be tapped from the furnace periodically. The inventor claims that the zinc oxide will not be contaminated with lead fume by reason of the conditions maintained in the furnace. (1,047,360, Dec. 17, 1912.)

Nickel-Copper Mattes.—A process for the separation of copper and nickel in high-grade or concentrated mattes, patented by Messrs. Alexander McKechnie, of Birmingham, and Frederic G. Beasley, of Smethwick, Birmingham, England, is claimed to be simple and effective. The raw or partly roasted matter is digested with sulphuric or hydrochloric acid at a

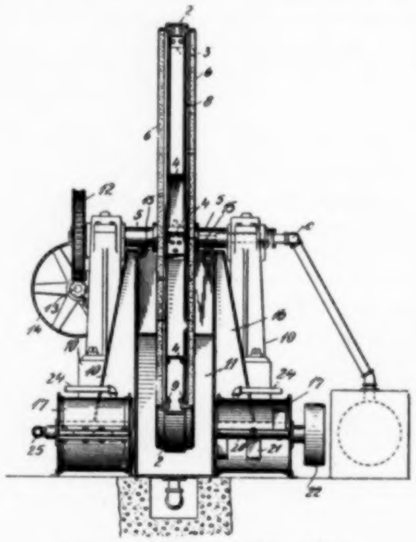


FIG. 4.—VACUUM SLIME FILTER

temperature of 100° C. in a closed air-tight vessel. Under these conditions of temperature and pressure the nickel is readily dissolved as nickel sulphate, when sulphuric acid is used, with evolution of hydrogen sulphide. The latter gas is utilized to the highest degree, under the conditions, for the precipitation of copper from solution as sulphide. Thus the single operation yields nickel in solution as sulphate and copper as a solid precipitate. Although the process can be applied to raw mattes, the inventors find that the action is accelerated by preliminary roasting. In the latter case, however, enough sulphide should be left to generate hydrogen sulphide for precipitation of copper. (1,047,825, Dec. 17, 1912.)

Rare Metals

Separating Uranium from Vanadium.—Solutions resulting from the treatment of carnotite contain both uranium and vanadium. The separation of these metals offers some obstacles, but may be accomplished readily according to a process patented by Mr. Warren F. Bleecker, of Canonsburg, Pa., and assigned by him to the Standard Chemical Co., of the same place.

The solution is first heated to about 90° C., and treated with heated sodium hydroxide in sufficient quantity to precipitate uranium as a mixture of sodium uranate and uranyl hydroxide. This precipitate carries some vanadium which must be separated by a further process. After filtering and washing the precipitate, it is dissolved in sulphuric acid. The solution is made slightly alkaline with sodium carbonate, and then electrolyzed with anodes of any desired metal, such as copper, nickel or iron. The vanadium will be precipitated as the vanadate of the anode metal, while the uranium will remain in solution and can be separated from the precipitate and converted into any desired form. (1,050,796, Jan. 21, 1913.)

Treatment of Radium-Bearing Ores.—A process supposed to be applicable to all known complex radium-bearing ores has been patented by Mr. Sidney Radcliff, of Bairnsdale, Victoria, Australia. It has for its object the separate recovery of the following products: (a) radium as radium and barium sulphates; (b) uranium as oxide or uranate; (c) acid earths such as tantalum, niobium, titanium, as oxides; and (d) rare earths such as cerium, thorium, lanthanum, didymium, etc., as oxides.

The ore or concentrate is crushed to pass a 30 or 40-mesh screen. It is then fused in a reverberatory furnace with about 2½ times its weight of acid sodium sulphate. When this charge has been fused, and while still fluid, sodium chloride to the extent of 10% to 15% of the weight of the ore is added and well rabbled. This causes complete decomposition and oxidation. The fused product is cooled and pulverized, after which it is agitated with warm water. Uranium, iron, the rare earths, and part of the acid earths will go into solution. Insoluble sulphates of radium, barium, lead and lime will remain in suspension. This turbid liquid is rapidly siphoned into a settler from which two products are obtained, namely, solution and fine slime.

The solution is treated with sodium carbonate, avoiding excess, which will precipitate iron, aluminium, chromium, uranium and the rare and acid earths. These are filtered from the solution and boiled with an excess of sodium carbonate which will cause the uranium to dissolve. The balance of the precipitate is treated with dilute sulphuric acid, whereupon all the slime except the acid earths will dissolve. These latter are filtered and ignited. The solution is treated with oxalic acid, which will precipitate the rare earths as oxalates, which are washed and ignited.

The fine slime contains most of the radium and is treated on the usual lines, and the crude radium and barium sulphates obtained in the usual way. (1,049,145, Dec. 31, 1912.)

Metallurgical Furnaces

Reverberatory for Copper Refining.—To prevent the disintegration and "floating up" of the bed of a reverberatory furnace, and to provide a furnace bed through which molten copper will not leak during the process of refining, are objects of improved furnace construction patented by Dr. J. B. F. Herreshoff, of New York City. In reverberatory furnaces hitherto used for copper refining the bed is composed of iron plates covered with fire brick on which is placed a covering of sand partly vitrified. A space beneath the plates constitutes a vault through which air may pass for the purpose of cooling the bed. It has been found that at times the molten copper will work its way down through this sand bed and cause the brick bottom to float, thus necessitating the temporary abandonment and repair of the furnace. The inventor has found that this disintegration of the bed is due to inefficient and imperfect cooling. As a consequence he proposes a construction whereby cooling may be accomplished uniformly, under suitable control, so that all parts of the bed may be kept cool enough to avoid the accumulation of molten copper below the brick bottom, and thus avoid disintegration and floating. In general, the invention may be said to consist of a series of pipes laid closely together in the bed of the furnace, both transversely and longitudinally, to carry air for the purpose of removing any desired amount of heat and thus keep the bed cool. Under this construction, any copper which may find its way through the bed will be cooled and solidified, and thus prevent further leakage. The heated air issuing from the cooling pipes may be used as preheated air at the combustion chamber of the furnace. (1,047,521, Dec. 17, 1912.)

Sintering Furnace.—In a patent issued to Mr. Milton H. Kauffman, of Denver, Colorado, the inventor describes the method of sintering or agglomerating fine mineral substances illustrated in Fig. 5. He makes use of removable pans 1 and 2, having perforated bottoms, which may be placed over an air box 6, which has a conduit 7 leading to an exhaust fan. His method of operation is to place a layer of material in the lower pan, with a layer of live coals in the upper pan, and set both over the air-box as shown. The exhaust fan draws air through the pans, as shown by the arrows, and ignites the layer of material in the lower pan. When ignition has proceeded far enough to continue without the aid of the live coals, the pans are removed and a third pan filled with mineral material is placed below the one previously ignited, and the pan containing burning fuel is removed. The exhaust continues to draw air through both pans with the result that ignition progresses from the upper pan to the lower, and the material in the upper pan becomes completely sintered. The operation is then repeated, placing new material in a lower pan, while the partly sintered material is placed above, and the completely sintered material is discharged. (1,050,079, Jan. 7, 1913.)

Zinc Retort Furnace.—Novel construction for introducing gas and air into gas-fired furnaces, is the basis of a patent granted to Mr. Alexandre Folliet, of Brussels, Belgium. In

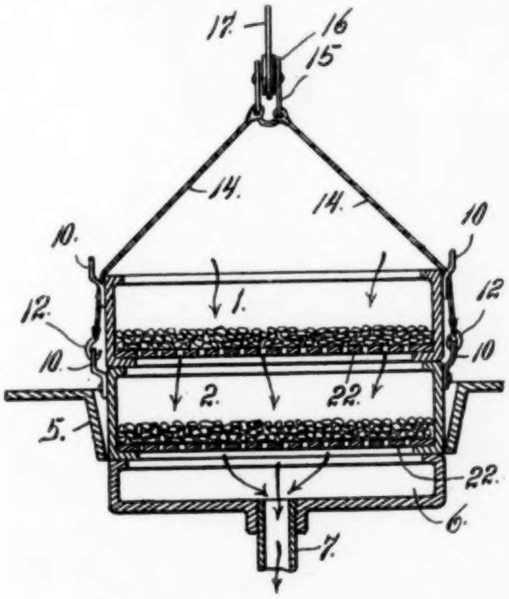


FIG. 5.—SINTERING FURNACE

furnaces of the type mentioned, it is essential that the distribution of heat be uniform both vertically and horizontally, and the object of the present invention is to improve the manner of mixing the gas and air used in combustion. Accordingly, the gas and air are introduced through separate, narrow, adjacent slits, constructed and arranged to cause the air and gas to enter the combustion chamber in the form of thin, vertically-directed streams or "slices," parallel and adjacent to each other. These slices will thus travel side by side, so that the contact is progressive, permitting of the maximum development of flame and radiation of heat, since the broadest sides of the slices are in contact. No opportunity is afforded for the mixture of air and gas prior to their introduction into the combustion chamber, thus avoiding improper mixtures and consequent formation of darts of flame even when air is forced by a blower. Each group of slits consists of one gas port between two air ports; or five ports or any other odd number may be used, the only requirement being that each gas stream should issue between two air streams. These groups of ports are disposed at suitable intervals along each half of the furnace. (1,049,569-70, Jan. 7, 1913.)

Chemical Industry

Manufacture of Cyanide.—The synthesis of hydrogen cyanide was accomplished by Berthelot by electrically heating a mixture of nitrogen, gaseous hydrocarbon and hydrogen, the latter being present in such excess that the quantity of hydrocarbon present did not exceed 10%. In this process the output was small, and not commercially profitable. Recently Ignacy Moscicki and Casimir Jablczynski, of Freiburg, Switzerland, have discovered that in the synthesis of hydrogen cyanide from the above constituents, the output increases with an increase of the percentage of nitrogen in the mixture, the proportion of hydrocarbon remaining the same. The dilution of the hydrocarbon with hydrogen, instead of with nitrogen, greatly reduces the output.

A specific example of the application of the improved process is given as follows: The basic substance is a gas made from oil having the following approximate composition: Methane, 50.3%; heavy hydrocarbons, such as acetylene, 22.5%; hydrogen,

23.2%, and nitrogen, 4%. This gas is mixed with pure nitrogen and hydrogen so that the mixture will contain 74 parts nitrogen, 10 parts hydrocarbon and 16 parts hydrogen. The mixture is preheated and treated in an electric furnace having a revolving high-tension flame. In this a part of the gas is heated to a temperature between 2500 and 3500 degrees Centigrade, and the elements form hydrogen cyanide. That part of the gas which was not highly heated in the flame is not decomposed, and issues with the hydrogen cyanide, cooling the latter and preventing its decomposition. That part of the gas which did not enter into the reaction is now replenished with the necessary ingredients and again passed through the furnace. The hydrogen cyanide formed may be dissolved in caustic solution to form alkali cyanide. An intermediate reaction in the process causes some of the saturated hydrocarbons to change into unsaturated hydrocarbons, thus liberating a quantity of hydrogen which, if not removed, would accumulate to such an extent as to greatly reduce the output of hydrogen cyanide. This removal may be accomplished by any one of several methods, namely, by diffusion, by oxidation, by catalysis with air, as a result of which additional nitrogen will be added; by burning the hydrogen with a deficient quantity of oxygen, nitrogen being again added to the mixture; by oxidation with copper oxide at a low red heat. (1,050,978, Jan. 21, 1913.)

Manufacture of Phosphorus.—A process for producing phosphorus from mineral phosphates has been patented by Mr. Frank S. Washburn, of Nashville, Tenn. The work of the inventor was based on the idea that a less volatile acid than phosphoric should displace the latter from its compounds, in a manner analogous to that in which sulphuric acid will displace hydrochloric and nitric from their compounds. After many experiments he found that it is necessary to mix a reducing agent with the phosphate and silica, and that the temperature of the electric furnace is required to cause the reaction to take place. When finely divided tricalcium phosphate, silica and carbon are subjected to the temperature of the electric furnace, the phosphorus is driven off rapidly and up to 90% of the whole. Sufficient air is brought in contact with the evolved phosphorus to form the pentoxide, which may be absorbed in any desired medium. (1,047,864, Dec. 17, 1912.)

Electric Storage Batteries

Preparation of Films or Flakes of Nickel.—In storage batteries of the Edison type, films or flakes of metallic nickel are used in the makeup of the positive electrodes. Mr. Thomas A. Edison, of Llewellyn Park, West Orange, New Jersey, has patented a method for producing these films, consisting in first making composite sheets of alternating thin layers of copper and nickel electrolytically deposited, cutting these sheets into strips and flakes of suitable size, and then dissolving the copper with a suitable solvent which will not affect the nickel. Such solvents patented by Mr. Edison are (1) a 25% solution of copper sulphate containing about 15 grams cupric chloride per liter. Other reducible haloids may be substituted for cupric chloride, but they are not as desirable; (2) a solution containing 285 grams ammonium sulphate and 10 grams cupric chloride per liter. When the metallic films of copper and nickel are agitated with these solutions, preferably warm, the copper separates as basic sulphate and may be decanted with the solution, leaving the nickel flakes to be suitably washed. This process is preferable to a former method of Mr. Edison's, in which he used an ammoniacal solution of copper sulphate, with attendant loss of ammonia, and the difficulty of recovering the copper in a desirable condition. (1,050,629-30, Jan. 14, 1913.)

Gage numbers of wire have no significance unless the gage is named. Thus No. 5 gage varies in diameter from 0.181 to 0.220 in. in six different standard gages. A far better way of specifying sizes of wire is to give diameter in decimals of an inch. This method is accurate and eliminate possibilities of error in filling orders.

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver

Air-Lifts vs. Centrifugal Pumps.—At the all-sliming cyanide mill of the Cia. Minera Jesus Maria y Anexas, San Jose de Gracia, Sinaloa, Mexico, centrifugal pumps were used to elevate the mill discharge and the final slime product. In the Colorado School of Mines *Magazine*, January, 1913, Mr. W. B. Rhodes describes the troubles encountered with the pumps, and their replacement with air-lifts. The pumps wore badly and gave constant trouble and expense. Of the two air-lifts installed, one had a 2½-in. lift pipe, with 65 per cent submergence, giving an effective lift of 18 ft. The other had a 3½-in. pipe, 54 per cent submergence and 38-ft. lift. The former was used for the sandy tube mill discharge and the latter for slime. The sand lift has plugged at times, but by the use of high-pressure air it has been cleaned without serious difficulty. Air is used at a pressure of 25 lb. per square inch. The total cost of the air-lifts, including the sinking of a shaft, 5 ft. square by 40 ft. in depth, was only $300. The cost of installing an old compressor was $150. The saving which has been effected by the air-lifts is as follows:

Maintenance of centrifugal pumps, including cost of pumps, liners, shafts and mechanics' wages, per month	$200.00
Saving of 20 per cent in horsepower, or 5 hp at $1.25 per day, 30 days	187.50
Saving 12 hours per month, equivalent to 40 tons ore at $25 per ton	1,000.00
Total monthly saving	$1,387.50
Total annual saving	$16,650.00

These two lifts have been working steadily for two years without interruption and have saved approximately $30,000. The air consumption varies from 2½ to 3 cu. ft. for each cubic foot of sand or slime lifted, depending on the dilution of the pulp. For the slime lift of 38 ft. a minimum dilution of 5:1 was found necessary to give good results. The sand is lifted at a dilution of 3:1.

Cyaniding at the San Poil Mill, Republic, Washington.—Preliminary cyanide tests on the ore of the San Poil mine indicated possible extraction as high as 98 per cent when the ore was crushed to 200-mesh and treated with a 5-lb. cyanide solution. The strong solution was required on account of the silver in the ore. Mr. Edward C. Morse describes the operation of the mill in the January, 1913, *Bulletin* of the Amer. Inst. Min. Eng. One of the interesting features of the work is the use of a Williams hammer pulverizer for crushing the ore to 3/16 or ¼-in. size. This machine receives the run-of-mine ore. It is similar in operation to the Quenner dry-placer machine used in Mexico for breaking cement-gravel; it has been used by the government for making sand for concrete purposes at the Celilo locks on the Columbia River. Its use in the San Poil mill is its first application to ore crushing.

The Williams hammer pulverizer consists of a central shaft carrying thirty-two 16-lb. hammers suspended by chains. The shaft revolves at 600 r.p.m. within a trommel built of grizzly bars 2 in. deep, with a 1-in. face, spaced 3/16 in. apart. The machine has shown a capacity of 15 tons per hour of run-of-mine ore. Sizing tests of the product have shown as high as 40 per cent passing 20-mesh, of which 65 per cent was finer than 10-mesh and 6 per cent as fine as 200-mesh. When the ore fed is wet the machine does about twice as much fine crushing as with dry material, but has lower capacity.

The crushed ore is sampled and run through a set of finishing rolls provided with carborundum blocks to prevent grooving of the rolls. The product is as fine as 8 or 10-mesh, and is sent to a tube mill, the discharge of which is classified by a Dorr classifier, and the sand returned for further grinding. The slime is thickened and agitated in a series of nine air-lift agitators, 12 ft. diameter by 18 ft. high. The tanks are connected in series, and the material passes from one to the

other by overflow. The agitated pulp is thickened and filtered in an Oliver filter, the filtrate being precipitated in zinc boxes. The barren solution is mixed with the discharge from the last agitator flowing into the thickener, thus giving the pulp a preliminary wash on its way to the filter. An extraction of from 93 to 94 per cent is made.

After several months' experience with the mill as described above, some changes were made. A new hammer mill was designed, heavier than the first one, and with grizzly bars spaced 1/4 in. apart. The rolls were replaced with a pebble mill, and it is intended also to install an additional Dorr thickener ahead of the filter.

Lead and Zinc

Lead Concentration.—In an article by Mr. R. S. HANDY in the January, 1913, *Bulletin*, Amer. Inst. Min. Eng., the author first gives a flow-sheet of the No. 2 unit of the mill of the Bunker Hill & Sullivan M. & C. Company, and then appends the following general considerations:

In this system there is a gradual reduction of the size of middlings by means of rolls as far as they are efficient, and, finally, by fine-grinding machines. At each step the freed galena is taken off, and the tailings disposed of before the material is again subjected to reduction. This complicates the process, but as the ore is hard it does not produce an excess of slime. The middlings are not mixed with the original feed until it enters the slime department, and the jigs on the original feed have a uniform load. The amount of circulating water is kept as low as possible by giving the launders ample grade to carry the products with a minimum of water. The fine trommels are washed with pipe sprays. The feed to rolls and grinding machines is first dewatered, and the wash water necessary to carry the discharge is applied through spigots with constant head from dewatering boxes.

Some of the newer machines, as far as the Cœur d'Alene is concerned, adopted in this mill, are the Chilean mill, Hardinge mill, Esperanza classifier, and Blake-Denison conveyor-weigher. Two Cooper Hewitt mercury-arc lamps are installed on the jig floor, and give remarkably distinct color values to quartz, quartzite, siderite, pyrite, blende, and galena, of which the ore is composed.

Some of the smaller mill details are: manganese steel roll shells trued with carborundum bricks; revolving trommel roll feeders; holes drilled in the bottom of fine-elevator buckets to destroy suction; the use of pure rubber washers between elevator buckets and the belt, to pass the fine sand instead of allowing it to lodge and wear the belt; air-lift for returning overflow water from the ore bins; the complete equipment of all bearings with Keystone grease cups with copper pins, and the use of double-edged or overflow boxes in the thickening tanks. The latter, while simple, are remarkably efficient, giving a perfectly clear overflow.

A complete system of sampling products and making tonnage tests of original ore, concentrates and tailings, makes it possible to give a complete report of operations up to any required point at any time during the month.

Macquisten Tube Flotation Process.—This process is in use at the Morning mine of the Federal Mining & Smelting Company, Mullan, Idaho, where Wilfley table tailings are separated into a concentrate containing zinc-lead sulphides and a gangue of siderite and quartz. In the January, 1913, *Bulletin* of the Amer. Inst. Min. Eng., Mr. O. B. HOFSTRAND describes the principles of the process and gives the details of treatment at the Morning mine.

The tube is a thin cylindrical shell, 3/16 in. thick, 6 ft. long, and 8.5 in. diameter. It has internal helical corrugations, preferably of quadruple pitch, advancing 3 in. with every turn around the tube, the pitch between corrugations being 0.75 in. It revolves in babbitted bearings, the feed being introduced at one end, and the products discharged at the other. The sulphide concentrates float and are discharged over a weir, while the gangue sinks and is caught in a receptacle at the end of the tube. As the ore advances through the tube it rises on the side of the tube and falls back onto the water, whereupon the sulphides float and are carried through in a thin film. The tubes revolve at 30 r.p.m., and ore remains in it about forty-eight seconds. The capacity depends solely on the carrying capacity of the water surface and the speed of surface flow, the latter being about 10 ft. per minute. Zinc concentrate can be floated at the rate of 175 to 200 lb. per twenty-four hours. The weight of lead concentrate would, of course, be greater. The range of size of particles amenable to this process is from 20-mesh to impalpable slime. The efficiency ranges, by actual tests, from 50 to 85 per cent. The feed, after being deslimed, is given a dilute sulphuric acid bath (0.05 to 0.5 per cent) to clean the particles. At the feed end of the tube there is added a small quantity of dilute soap solution and oil, to increase the surface flow carrying the mineral particles. The cost of treatment based on a capacity of 100 tons per twenty-four hours, in-

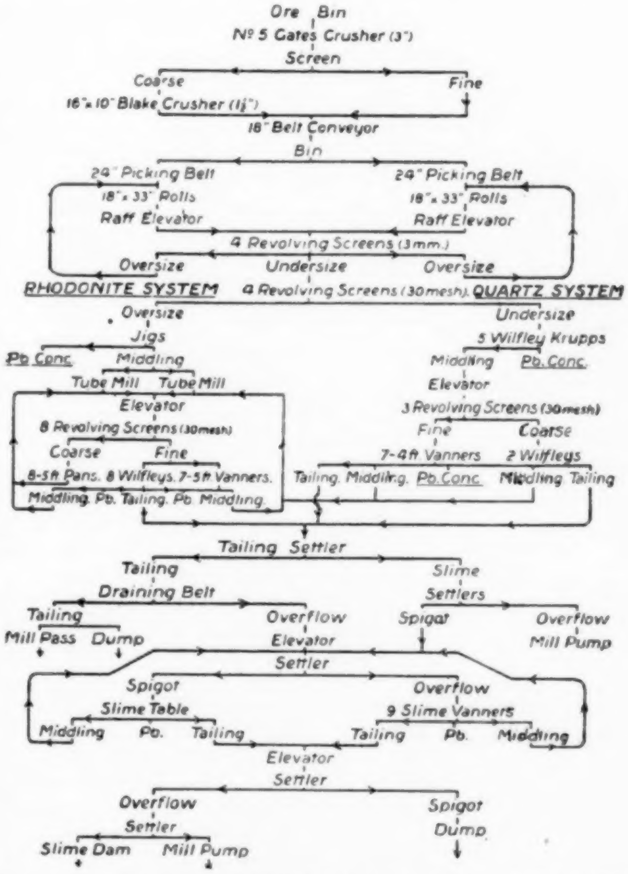


FIG. 1.—FLOW SHEET, DECEMBER, 1911, JUNCTION NORTH MINE, BROKEN HILL, N. S. W.

cluding cost of acid, oil, power, attendance, repairs, and supplies, is approximately 40 cents per ton, using four tubes in series. The actual cost for acid, soap, and oil at the Morning mine is less than 1 cent per ton of ore treated.

Lead-Zinc Concentration in Australia.—The accompanying flow-sheet, Fig. 1, shows the system adopted at the Junction North mine, Broken Hill, N. S. W., in the treatment of Broken Hill lead-zinc ore carrying more rhodonite than usual in that district. The details of the system are described by Mr. STANLEY C. BULLOCK in *Bulletin* No. 100, Inst. Min. & Met. Following is the analysis of an average sample of the ore over a period of one year (see Table I, p. 164).

Table II (p. 164) gives a résumé of the work done in the mill during a week in February, 1911.

Three years ago this mill was one of the worst equipped in the district; its output was a little over 2000 tons per week, the lead concentrate rarely exceeding 64 per cent lead, while the tailings often was as high as 6 per cent lead. Subsequently the capacity of the mill was increased and its work improved by adding more screens, grinding apparatus, and tables. The di-

Table I

	Per cent.
Insoluble	55.35
Lead	14.61
Zinc	8.65
Iron	5.17
Sulphur	7.63
Peroxide of manganese	5.70
Lime	1.03
Alumina	0.57
Carbonic acid	1.05
	99.76
Silver	10.3 oz. per ton.

Table II

	Tons	Assay Value			Per Cent. of Crude		
		Pb %	Ag oz.	Zn %	Pb %	Ag oz.	Zn %
Concentrate	529	68.4	35.1	4.0	71.1	50.5	7.5
Tailing	2052	3.9	5.4	9.6	15.7	29.4	69.8
Van tailing	292	10.0	12.4	13.8	5.7	9.8	14.11
Slime to dam	185	19.0	19.0	12.4	6.98	9.6	8.13
Slime to storage tank, 8 % H^1O	8	33.8	23.2	11.6	0.52	0.5	0.32
Mine Assay	3225	15.4	11.4	8.7	100.00	99.8	99.86

vision between quartz and rhodonite treatment also was adopted.

Power is supplied by a 175-hp high-speed engine and partly by a 250-kw generator. The consumption on the different machines is given below:

STEAM

	Speed	Tons per Hour	Horsepower	
			Each	Total
2 Rolls 33 in. by 18 in.	24 r.p.m.	10.5	30	60
4 Elevators 12 in. by 6 in., 40 ft. centers	365 "	10.5	9	36
17 Wilfley tables	260 "	0.75	1	17
14 Vanners 5-ft	240 "	0.75	½	7
4-ft	240 "	0.5		
1 Elevator 20 ft. centers	400 "	5.5	6	6
1 3-in. Centrifugal pump for lead	1200 "	0.5	6	6
1 5-in. Circulating water pump	1000 "	.....	6	6
16 Revolving screens	10 "	5.25	⅛	2
1 jig	200 "	10.5	2	2
Lead conveyors	54 ft. p.m.	.....	2	2
Tailings draining belt	35 "	16	2	2
Small elevator 6 ft. centers	350 "	.....	2	2
4 Feeders	.....	.....	½	2
Total steam power				150

ELECTRIC

	Horsepower		Tons	Total H.P.
	Max.	Av.		
1 5D Gates crusher	35	25	48	25
1 16-in. by 10-in. Blake crusher	25	15	40	15
1 Crude ore conveyor	15	12	48	12
2 Tube mills	35	25	10	50
8 Grinding pans	35	7	½	56
1 Vanner section	25	22	....	22
2 Pumps	16	16	....	32
1 14-in. conveyor	4	2.5	16	2.5
Total electric power	.....	.....	.....	214.5

The average cost of steam power is 2 cents per hp-hour, and of electric power 1½ cents per hp-hour. The mill is treating well over 3000 tons of crude ore per week of 144 hours.

Some points of interest in the mill are the following: All Wilfley tables are fitted with riffles ending in a special curve at the delivery end, determined by experiment, to separate the fine lead from the rhodonite. All launders are of ⅛-in. steel, parabolic in section. A 3-in. centrifugal pump with flooded suction and 2-in. delivery pipe, gives satisfaction in raising lead concentrates at the rate of 1000 lb. per hour. The draining belt used to carry tailings from the mill is 24 in. wide and 50 ft. long. The first 20-ft. section is in the form of a semi-circular trough, with slight slope so that water will drain back. The belt then rises sharply at an angle of about 15 deg., and this, with the addition of a bumper, dewaters the tailings sufficiently for use in underground filling. The moisture can thus be reduced to 5 per cent.

A minerals separation plant is now being erected to treat the ore after the jig lead has been extracted. The mill circuit will be broken at the table screens in both quartz and rhodonite sections, and from these all the undersize will run to the flotation plant. Old dump material also will be treated by flotation.

Chemical Analysis and Assaying

Variations in Assaying Gold Ore.—Tests were made at the Akaska-Treadwell mine to determine the error in assaying gold ores, and the degree of accuracy which would be considered satisfactory for mill-feed, concentrate and tailing. Mr. W. P. Lass gives the details of the experiment in the *Mining Magazine* for January, 1913. To summarize the results, the average variation of one assay from another on mine and mill samples, averaging $3.46 per ton, was 41.3 per cent. The average variation on tailing samples, averaging 32.7 cents per ton, was 20.6 per cent. The average variation on concentrate samples, averaging $68.79 per ton, was 7.6 per cent.

The author accounts for these variations by the fact that the gold occurs in Treadwell ores in free gold particles. There are at least 280,000 particles in one assay-ton of 40-mesh sand. If one of these particles be fine gold it would increase the gold content of the ore $21.52 per ton. In one assay-ton of cubic particles of the same dimensions as the openings in a 100-mesh screen there are at least 5,348,811 particles; if one of them be fine gold it would increase the assay result $1.12 per ton.

In ten different samples of concentrates the average first assay was $68.81; the average second assay was $68.77; the average variation was $5.21, or 7.6 per cent figured on the average of twenty assays and the average variation. Twenty-eight mine and mill samples gave an average first assay of $3.50; average second assay $3.41; average variation, $1.43, or 41.3 per cent figured on the average of fifty-six assays and the average variation. A point of particular interest is that though the average variation may be great, the average value of the samples remains approximately the same. It is a compensating variation. In ten tailing samples the average first assay was 32.5 cents per ton; the average second assay, 32.9 cents; the average variation being 6.7 cents, or 20.6 per cent figured on the average of the twenty assays and the average variation.

Assay of Cyanide Solutions.—The different schemes proposed for the assay of gold-bearing cyanide solutions are reviewed by Mr. L. J. Wilmoth in the February, 1913, *Mexican Mining Journal*. The following diagram shows the classification of the different methods:

Cyanide Solution

- Evaporation
  - Lead Cups
  - With Soda
  - With Litharge
- Precipitation
  - Electrolytic
    - Arent
    - Chiddy
    - Holt
  - Chemical
    - Crosse
    - Whitby
- Calorimetric
  - Moir
  - Cassel
  - Prister

Of the evaporation methods, it is the opinion of the author that the first is antiquated and no longer in use; the second is seldom used, and is inferior to the third, which he regards as undoubtedly the best, but undesirable on account of the time needed.

Of electrolytic precipitation methods, there are those in which the gold is deposited by means of an extraneous current, and those in which the gold is deposited by means of an electric couple formed by two metals brought into the solution to be assayed. The first method is unimportant and has never been brought into commercial use. The second class of methods includes several which are of interest and importance. In

all these methods a given quantity of cyanide solution is acidified with sulphuric or hydrochloric acid; a solution of copper sulphate or lead acetate is then added, and the gold precipitated by introducing metallic zinc or aluminium to form the electric couple. Arent uses sulphuric acid, copper sulphate and aluminium. Chiddy's method requires hydrochloric acid, lead acetate and zinc shavings; and Holt proposes the use of hydrochloric acid, lead acetate and aluminium. The author believes the method of Chiddy to be the best; it is most extensively used. The use of copper solution is undesirable on account of the difficulty of eliminating the copper in the assay. Zinc is preferable to aluminium as it passes into solution more readily, is cheaper, more easily procured and just as effective.

In the second division of precipitation methods we find those of Crosse and Whitby. The former uses an excess of silver nitrate to precipitate a double gold-silver cyanide, while the latter relies on an acidulated solution of copper sulphate to throw down a double gold-copper cyanide. Crosse's method is regarded as costly, and has been superseded by others. Whitby's method is regarded as being in the same class with Chiddy's and the evaporation with litharge is being most extensively used.

The calorimetric methods have not been widely adopted. They all depend on the precipitation of gold from solution, redissolving it and then adding stannous chloride to form purple of Cassius. Moir first destroys the cyanide by addition of sodium peroxide. A few drops of lead acetate solution is added and a very small quantity of aluminium stirred in. This is not unlike Holt's electrolytic method. Prister precipitates the gold with a solution of copper sulphate and an alkaline sulphide, while Cassel uses potassium bromide and sulphuric acid.

Of all the methods proposed there are only three that are widely used. The evaporation method may be said to be in universal use; in America Chiddy's method is widely adopted, while Whitby's method is used in most of the mines of South Africa.

Assay of Zinc-Dust Precipitate.—In a paper presented before the American Metallurgical Society in June, 1912, Mr. H. R. LAYNG gives his procedure in the assay of zinc precipitate. The sample is ground on a bucking board to pass a 60-mesh screen. It is then thoroughly mixed and finally spread to a uniform depth of about ½ in. By the use of a square-end spatula a sample is taken by dipping from various parts of the layer of dust. This is dried, cooled, and then mixed and spread again. Three samples of 0.1 A.T. each are then dipped by means of a spatula 1/16 in. wide. Each sample is placed in a 10-gm. Battersea crucible, previously dried and containing 10 gm. test lead and 1½ gm. of a mixture composed of one part each of potassium carbonate and sodium bicarbonate and two parts borax glass. The precipitate is mixed with this charge and the crucible is tapped to settle the mixture evenly.

Three fluxes are used in assaying the original charge, the resulting slag and the cupels. They are shown below:

	No. 1. gm.	No. 2. gm.	No. 3. gm.
Sodium bicarbonate	400	600	1 part
Potassium carbonate	400	....	.... 60 gm.
Litharge	800	1200	5½ parts
Silica	250	200	8½ gm.
Charcoal	30	12½	1 gm.
Borax glass	...	....	3 gm.

After mixing the sample with lead as described above, 20 gm. of flux No. 1 is added, allowing it to wash around the sides of the crucible. This is leveled and covered with 20 gm. of flux No. 2. The general effect of these fluxes is to catch any gold or silver which might be carried to the top of the crucible. It will be noted that the charge is of such a character that lead is reduced in successive stages, and not all at once as in the ordinary assay. The crucibles are subject to low red heat until fused, then finished at a higher temperature for 10 minutes, and poured.

The buttons are cupelled carefully. The slag is assayed in the original crucibles, using 20 gm. of flux No. 2 and ⅜ gm. charcoal. The button resulting is cupelled separately, and the weight added to that of the original.

The cupels are assayed for absorption loss by grinding to 150-mesh, and assaying 10 gm. with 70 gm. of flux No. 3, together with 1 gm. charcoal and a borax cover. The author finds that his method gives higher results than other fusion methods, and compares favorably with the volumetric methods for silver. Assays check within 5 or 10 oz. on precipitate carrying 20,000 to 22,000 oz. silver.

Electrolysis

Chlorination of Benzene by Electrolysis.—A paper by **R. G. Van Name** and **Carlton H. Maryott,** in the February, 1913, issue of the *American Journal of Science* discusses the mechanism of the chlorination of benzene in the electrolytic cell. The investigation was carried out in the Kent Chemical Laboratory of Yale University. The chief results are as follows: (1) Electrolysis of benzene in a solution of lithium chloride in glacial acetic acid gave chlorinated benzenes with current yields, under favorable conditions, of 50 to 70 per cent. Both addition and substitution chlorine were present in the product, the latter predominating. There was, however, nothing to show that the effects were not due to the secondary action of chlorine previously set free by the current and dissolved in the liquid.

(2) Benzene dissolved in the same solution is really chlorinated by direct treatment with chlorine gas in the dark, yielding addition and substitution compounds in proportions which vary with the conditions.

(3) The addition of benzene to the acetic acid-lithium chloride solution during electrolysis raised the anode potential, and by an amount equal to or greater than that produced by a similar amount of carbon tetrachloride. This indicates that benzene has little or no depolarizing power toward chlorine, at least in this case. Phenol, added under similar conditions, produced a decided lowering of the anode potential.

(4) The rate of chlorination of benzene, when dissolved in the same medium saturated with chlorine, was measured at 10.2 deg. Simultaneous electrolysis did not accelerate the chlorination perceptibly under conditions where a 10 per cent current yield should have been evident. The current yield, if appreciable, was therefore below 10 per cent.

(5) The products of 4 contained addition and substitution chlorine in the ratio of about 2 to 3, this ratio increasing slowly as the reaction progressed. The reaction velocity, calculated on the assumption that monochlorbenzene and benzene hexachloride were the sole products, showed a steady rise, but the rate of chlorination of monochlorbenzene, separately determined, appeared to be sufficient to account for this effect.

(6) No positive evidence of strictly electrolytic (*i. e.*, anodic) chlorination of benzene was obtained.

Non-spinning wire rope, the invention of Olof Tangring, an engineer of the American Steel & Wire Company, is made by first winding one series of wire strands over a core in one direction, and an outer series over the inner series as a core in the opposite direction. The wires composing the inner strands are twisted in the same direction with them, but in the outer strands the wires are laid in the opposite direction to them. The construction results in the counteraction of the tendency of the rope to spin when used in hoisting or lowering loads.

The melting point of cupric oxide has been investigated by R. E. Slade and F. D. Farrow (*Zeit. Electrochem.*, XVIII, 817), who find that the pure salt does not melt below 1148 deg. C. Decomposition occurs at this temperature, even with a pressure of 2.5 atmospheres, producing cuprous oxide which forms a solution with the cupric oxide solidifying at about 1081 deg. C. Under a pressure of one atmosphere the decomposition begins at 1090 deg. C., and the solution, containing over 50 per cent cuprous oxide, solidifies at about 1064 deg. C., which is the value given by Wöhler for the true melting point of copper oxide.

Chemistry in Russia in 1912

Russia as a chemical producer is a comparatively new comer. But there are some branches of the industry which have acquired what the Russian calls "citizen rights" in the country and which have developed in a very satisfactory manner. These are particularly soda and acids.

As far as the year 1912 is concerned the year has been a very satisfactory one, although more from the importers' point of view even than from the manufacturers'. The Russian field for foreign chemical products is an immense and a growing one, and will be worth more attention for many years to come, because the industry, as suggested above, is only partially developed in the country. Anyway during the year under review the factories were kept working all the time, and some of them were at times in straits to comply with the orders they received.

The market value of chemicals in the country did not vary very much last year, but there are some leading items that suffered changes in price, such as sulphuric and nitric acids, borax, prussiate of potash, and the like. Prussiates rose rather briskly, because of the stocks becoming exhausted both in Russia and Germany. There was also a substantial advance in lead, zinc, and copper compounds on account of the increase in the value of the respective metals, and the advance was maintained right to the end of the year, or nearly so. But the important Russian chemical products, as stated, are sodas and the value thereof had slightly declined in comparison with 1911 both for calcined and caustic. But it is objected and regretted that these particular goods should be so high in prices when they enter so largely into the economic life of the country. For example, the average cost of calcined soda last year was 1 rouble 16 copecks per pood; and the average cost of caustic soda was 2r. 47c. per pood.

Chemicals as a rule are protected by a tariff, and it is only when an agitation grows pretty strong that the quasi monopolists consent to make a slight reduction in order to avoid further attention being called to the fine business they are doing. But perhaps the most interesting development in the Russian chemical industry is the most recent one, namely, the alacrity with which the South Russian metallurgical works are adopting byproduct coke ovens; the most interesting for the moment among the byproducts is ammonia, because the Russian agriculturist is becoming a larger consumer every year of fertilizers—both home-made and imported.

But although this business has been catered to generously in the South, the first cost of the soda appears to the agriculturist to be so high as to discourage him from adopting it on a large scale; and the consequence has been that the production of sulphate of ammonia has not made the progress that was expected of it. It will be some time evidently before the Russian farmer will learn to appreciate the value of ammonium sulphate as a fertilizer. On the other hand, he is becoming a very large consumer of superphosphates. These were imported last year in far greater quantities than ever before—in fact, the imports increased at a much greater rate than the production at home, which is not unimportant. It is not that phosphate rock does not abound in Russia that causes him to buy the superphosphate abroad. Phosphate rock is plentiful enough in the country; but the difficulty is to get sulphuric acid cheap enough; and the phosphate rock is not found in comfortable proximity to either native sulphur, of which there is a very little in the country, or pyrites.

In the Urals region there are large deposits of phosphate rock close enough to the pyrites; but unfortunately these particular deposits are poor and it becomes a question whether it will pay to work them even with convenient supply of acid. On the other hand, the Zemstvo (that is, district council) of Perm has decided to erect a superphosphate factory at public cost in order to supply the agricultural needs of the Perm and Viatka governments. The richest phosphate rock deposits in Russia are in the Southwest (Podolia) and they are largely exported into Austria unmanufactured. There does not appear to be sufficient enterprise amongst the industrial chemists of Russia to turn these deposits to account. The trouble is always, they will tell you, the question of acid.

The importation of fertilizers into Russia in 1912 during the first 10 months amounted to close on 24,000,000 poods, which compares with a little over 19,000,000p. in 1911 and close on 16,000,000p. in 1910, always during the first ten months.

It is hoped that in the course of time Russian industrial chemistry will be able to deal with the other byproducts of the coke oven, such as benzol and so on, for the purpose of establishing an independent color industry, but that time is probably still far off.

Besides superphosphates, the Russian farmer consumes large quantities of Chile saltpetre and German potash salts, both of which were imported last year in larger quantities than in any preceding year.

While Russia manufactures largely for her own use under the protection of a high tariff wall in several branches of chemistry, she can hardly be said to have an export trade in any of them, an exception being the products of wood distillation. This industry in Russia has entered into a very interesting phase. For many years back Russia has exported a turpentine of deplorably inferior quality which only gets a chance, say, on the English market when American turpentine becomes very high in price. A large customer for it is Germany where it is hardly ever turned to any account, however, until it has passed through the refineries there. But it is used in other countries, and perhaps in Germany, too, for mixing; but probably not to any large extent.

During the last year or two systematic experiments have been made for distilling the resin and it is claimed that this has been accomplished wth great success, the prospect held in view being that in a very few years French, and particularly American, turpentine will have lost their significance for the Russian market; because the Russians have the raw material in abundance, there will be no difficulty in supplying their own needs independent of supplies from either America or France to which countries they have had to look in the past for turpentine of anything like quality. This applies also to rosin.

There is a point, however, that has to be considered in connecton with this feature of Russian chemistry, namely, the question whether the Russian climate will be favorable to tapping the trees, as is done in France. Should it prove that the severity at times of the Russian weather does not discourage this preliminary operation in connection with the production of turpentine and rosin, it is quite likely that American and French goods under these headings will soon cease to be quoted on the Russian market.

As hinted in the first few lines although Russia has made enormous strides in the manufacture of a limited number of chemicals, if we take the whole field she is still so far behind in the race that chemical manufacturers in other countries may well cultivate the market in full confidence that it will available for their goods for many years to come. Germany knows this and is doing an immense business with her neighbor over the border, not only in colors, in respect to which, of course, she leads the world, but also in other chemicals when other countries may easily become serious competitors.

St. Petersburg, Russia.

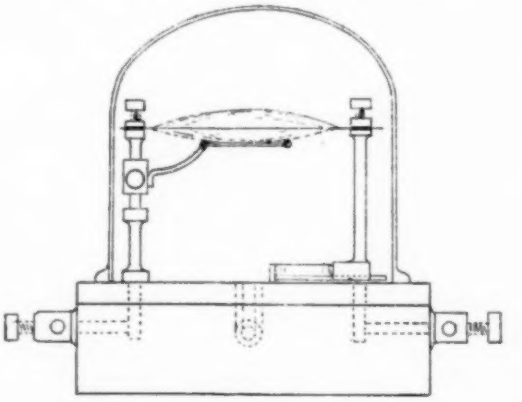
Six **Chapman Rotary Gas Producers,** built by the Chapman Engineering Company, of Mount Vernon, Ohio, are being installed at the open-hearth plant of the Bethlehem Steel Company at South Bethlehem, Pa. Three Chapman gas producers of the same type have been ordered by the Tremont Nail Company for its open-hearth at West Wareham, Mass., and one by the Crucible Steel Company, of America for its plant at Syracuse, N. Y.

The consulting engineering firm of Mr. **E. S. Lincoln,** of Waltham, Mass., has been incorporated as E. S. Lincoln, Inc., and has moved its offices to the new laboratory now nearing completion at 129 Bacon Street, Waltham.

The Apophorometer

A paper by Prof. **J. Joly** in *Philosophical Magazine*, February, 1913, describes the "apophorometer" which is an instrument for enabling sublimates obtained from substances at high temperatures to be collected in their entirety and weighed on the chemical balance for purposes of chemical analysis, identification or research.

The apophorometer is a very simple instrument. A ribbon of this platinum, about 6 cm. in length and 4 or 5 mm. in width is stretched between two forceps which are provided with binding-screws so that a current can be sent through the ribbon, raising its temperature to any desired degree up to the melting point of platinum. One of the forceps is movable parallel to itself, and is so acted on by a light spring that the ribbon is kept stretched. The substance to be volatilized is placed in the form of powder upon the ribbon or hob. Beneath the ribbon a watch-glass is held in position by a support which can be raised or lowered or rotated to one side. When this watch-glass is in contact with the ribbon a second watch-glass is placed on the lower one. The ribbon is thus enclosed between the glasses for nearly its entire length. Overall a receiver fits enabling



APOPHOROMETER

an indifferent gas to be introduced around the heated substance or a vacuum to be established.

The procedure is as follows: The lower watch-glass is raised till it is in contact with the ribbon. From 5 to 30 milligrams of the substance are distributed over the ribbon so as to extend nearly over its entire length. The upper glass is then laid on. All is now ready for the experiment. In some cases it is desirable to weigh the platinum ribbon before placing it in position. And again in some cases it is advisable to make a preliminary experiment on a portion (not weighed) of the powder in order to observe the behavior of the substance.

A current is now passed through the ribbon and gradually increased. It may be that more than one sublimate is obtainable from the substance by careful regulation of the temperature; or it may be that only the one is to be expected, and the residue being non-volatile very great care in raising the temperature is unnecessary. When the sublimate is coming off freely the high temperature is maintained unaltered for from 10 to 15 minutes or until the sublimation is seen to be completed. The current is then cut off and the glasses let cool; but while still considerably above air-temperature the upper glass is lifted off, the lower one let down and removed, and both glasses are clipped together for weighing. This is important, as some sublimates are hygroscopic. It is well to use watch-glasses which are ground to meet accurately.

It has been assumed that the presence of oxyen is not injurious. In many cases it is not. If it is, the receiver must be taken into use and the air excluded.

If there is any uncertainty as to the completion of the process, a second pair of watch-glasses should be placed around the ribbon and the current again turned on. We can thus make sure that the volatile constituent has been completely removed.

The whole process is one of ease and simplicity. In most cases the nature of the sublimate is anticipated, and in others its character enables it to be determined; and, of course, chemical tests may be applied to it after its weight has been ascertained. Knowing the nature of the sublimate, the combined oxygen may be calculated out.

The weight of residue left upon the ribbon is in some instances of importance. But it is necessary to bear in mind that the residual element may be oxidized in those cases where access of oxygen has been permitted. It will sometimes be found that from this cause the residue may actually weigh more than the original sample of the substance.

Certain substances may alloy with the platinum, and, owing to the extreme thinness of these wide ribbons, cause it to fuse. These may be dealt with either on heavier platinum ribbon or on carbon. Molded strips of carbon would, doubtless, be obtainable. In point of temperature-range carbon has also advantages, but this is of minor importance, for there are few sublimates not obtainable by the use of platinum. It should be remembered that ordinary carbon may contain volatile impurities. If we desire to deal with very high temperatures, the watch-glasses should be of transparent vitreous silica. The results obtained by this method with various substances are described.

The Non-Ferrous Metal Market

In general the tone of the non-ferrous metal market is weaker than reported a month ago. In view of the unsettled conditions, buyers are not anxious to enter the market, and as a consequence concessions have been made by the sellers. Special brands have commanded special prices, but ordinary requirements have been met at lower figures.

Copper.—Concessions have been made to induce buyers to purchase, and the market has fallen to a point that makes buying attractive. Few transactions are reported in Lake copper, and the quotations are nominal. The latest quotations are, for Lake, 15⅝@15⅞ cents; and for electrolytic, 15.25@15.50 cents.

Tin.—The market is generally lower in view of the decline in London. Domestic purchasers have entered the market freely and the metal is again becoming scarce. February tin in the domestic market is quoted at about 49⅜ cents.

Lead.—This market is quiet, and prices have not changed materially. The foreign market is slightly firmer. The latest quotations are 4.15@4.20 cents, St. Louis, and 4.30@4.35 cents, New York.

Spelter.—This market is weak with a slight demand. The price has declined twice during the month, and at last reports was 5.05@6.15 cents, St. Louis, and 6.20@6.30 cents, New York.

Other Metals.—The aluminium market has been rather dull, with metal selling at 25¾@26 cents, New York. Prices for antimony also have declined slightly, different brands bringing from 8.25 to 9.60 cents per lb. The quicksilver market is quiet but prices are unchanged at $40 for domestic orders and $37.50 for export.

Portland cement manufactured in the United States in 1912 is estimated by the U. S. Geological Survey at 81,941,998 barrels, which estimate is believed to be correct within 1.5 per cent. The production showed an increase of 4.3 per cent over that of 1911. The curve of production of Portland cement is still rising and has yet to take its first downward turn.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in November, 1912, was 62. The total quantity of ore milled during that period was 2,203,900 tons. There were 9966 stamps in operation with an average duty of 8.58 tons per 24 hours. Tube mills in commission numbered 281. The yield for the month was 757,337 fine ounces gold. The report shows a steadily decreasing number of stamps and increasing number of tube-mills in use.

Standard Screens for Screen Analyses

By Robert H. Richards

Great interest has been shown in recent years, and rightly so, in the importance of having standard sizes of screens, or sieves, in laboratories where sampling and screen-sizing has to be done. The need of such a standard is occasioned by the fact that without it the worker on mill or furnace products in one part of the country can not read the article written by the worker in another part of the country, and translate the figures obtained into the measures that he is used to in making his own experiments, unless he knows the sizes of grain that are dealt with in the experiments described.

Some efforts have been made in this direction up to the present time, and the writer will refer to two. The sieve scale adopted by Rittinger, in his work on ore dressing, started with 1 mm. and laid out the different sizes for the screen holes, for sizes above 1 mm., by multiplying each lower size by 1.414, or the square root of 2, in order to obtain the size next larger; going up in this way until he reached the largest size that in all probability would be needed, namely 64 mm., which is about 2½ in. in size, and going down to sizes below 1 mm., by dividing the size of the hole in each screen by 1.414, in order to obtain the size of the hole in the screen next smaller, and so on down the series until the smallest possible size of screen hole was reached. In bringing out this sieve scale so far as we know Rittinger simply worked it out as a very desirable plan, but so far as we know no manufacturer took up the making of the screens, and no standard set of screens were to be had.

The second instance to which the writer will refer is the standard set of sizing screens adopted by a commission appointed by the Institution of Mining and Metallurgy of London, and presented at their twentieth session 1910-11. This commission took the ground that miners, mill men, and metallurgists are so wedded to the use of the word "mesh" as description of the size of a sieve that it was unwise to try to introduce any other mode of designation except "mesh."

It is known to all who have given any thought at all to the matter that the word "mesh" does not define the size of the sieve hole, and therefore the size of the product which the sieve makes; for example, the writer has before him the catalog of a manufacturer of screens, and in it he sees that a 14-mesh screen, with the coarsest wire that is ever used by this company would have a hole 0.030 of an inch in diameter, while the 14-mesh screen, with the finest wire that is ever put into that size of screen would have a hole 0.061 of an inch in diameter; and there are sieves made by this manufacturer that have fourteen different sizes of wire, ranging between the above two extremes; 14-mesh sieve therefore has sixteen different sizes of hole according to the sixteen different sizes of wire used. Therefore the writer who speaks of a 14-mesh sieve in his descriptive article may mean to his reader that he is talking about grains anywhere from 0.030 of an inch to 0.061 of an inch, failing utterly to convey a scientific fact to his reader.

The commission, aforesaid, in making their screen set, known as the I. M. M. standard screens, fully realized this difficulty, and met it by arranging to use wires of the same diameter as the width of the holes. This gives the reader definite information as to the size of the grain that is meant when a particular mesh of the sieve is used. These screens, however, do not have a constant ratio between two adjacent sizes; in fact the ratio between the width of 100-mesh holes and 90-mesh holes is 1.095, while the ratio between 30-mesh and 20-mesh is 1.508. Other lesser variations occur all the way along the series, and in fact there is no attempt at a uniform ratio.

There is another difficulty which this method runs against, and that is, that in order to have screens of uniform size of opening, we must use double-crimp wire cloth, and a prominent manufacturer of screens informs the writer that it is quite difficult to make double-crimp wire cloth where the wire is as large as the size of the holes, particularly when one gets to the coarser sizes, and if single-crimp wire cloth is used, the wires are not fixed, and the sizes of holes will vary to a ruinous extent by the slipping of the wires.

In regard to the use of a constant ratio between the sizes of holes of the adjacent sieves, the writer believes there are very strong reasons in favor of using a constant ratio. If one is making a series of sizing tests from a crushing machine, one wishes to know the proportional amount of the sizes at different points down the series, and will probably wish to draw a curve of the same. If one pair of sieves has a very small ratio of sizes of holes, and another has a very large ratio, then no regular statement of quantities of products made by the oversize of the successive sieves will give its full meaning, nor will it be possible to make a curve which will truly represent the result of that crushing operation.

Also if the series of products obtained are to be subjected to any concentrating operation, with a view to ascertain the ability of the ore to be concentrated, there will be no fair comparison between two products, using the above illustration, where in one instance the coarsest grain in the product is 1.5 times the size of the finest, while in the second case the coarsest grain in the product is only 1.09 times the size of the finest grain in the product. The work of concentrating on two products differing so complctely in their limits of size as those two products would differ, would fail entirely of being comparable one with the other.

Again, the very fact of using "mesh" as the standard of size of sieve invites the man who has not obtained the I. M. M. standard sieve to talk about his products by use of the word "mesh," when his products are made by sieves of different make, and therefore have different sizes of holes from the I. M. M. standard screens.

On these three accounts (1) the difficulty of making double-crimp wire cloth of such coarse wire; (2) the difficulty of drawing deductions from curves, and of comparing results; and (3) the difficulty of having others not provided with the I. M. M. standard entering into the discussion and using the word "mesh," it seems to me that a standard set of screens, which is standardized according to the size of its hole, instead of according to the number of meshes, and that has a constant ratio from one end to the other, for example, Rittinger's ratio of 1.414 or the square root of 2, and having the liberty to use wires that are smaller than the size of the hole, admitting therefore of the use of the most perfect kind of double-crimp wire cloth, can not fail to have great advantages over the series of standard screens based upon mesh.

Such a set of standard screens has just now been prepared by the W. S. Tyler Company of Cleveland. This set of screens begins with 200 mesh as its starting point, which is a size of screen that has been standardized at Washington by the U. S. Government, and uses the ratio upwards from that point of 1.414, running all the way to the maximum size. These screens are made with the greatest care in order that they shall always be standard in size of hole, and in order that the various experimenters in the different parts of the country who make screen analyses of their products can read each other's results, and know exactly what is meant by the results.

Since the ratio between the width of the hole between the next small screen and the next larger is always the same, namely, 1.414, a sizing test made by this set of screens will give the proportional quantity of material that belongs between each two sieves according to a perfectly definite law, that is to say the largest grain in any product will always be 1.414 times the size of the smallest grain in that product. As a consequence of this the series of products obtained will have their quantities, or weights, proportional to a perfectly definite law, with the variable ratio entirely eliminated. They will yield curves which truly represent those quantities according to that definite law, and if concentrating tests are desired upon these products, the results of concentrating will be comparable all the way down the line of products, ranging from the coarsest to the finest.

The writer warmly commends this effort to make a valuable standard set of screens for use in screen-sizing tests, and hopes that they will generally be adopted by the mining engineers, the mill men, and the metallurgists of the country, in order that the papers written on these various subjects may have the greatest uniformity in the description of the products entering into such discussions.

Massachusetts Institute of Technology,
Boston, Mass.

An American Oil Engine Test

The Busch-Sulzer Bros.-Diesel Engine Company of St. Louis, Mo., have just issued a well illustrated booklet, entitled "The Diesel Engine—A Test," which seems to deserve some comment.

The booklet gives the report of an exhaustive test of a 225-hp, 4-stroke cycle, Busch-Sulzer-Diesel heavy oil engine. This is the engine invented by Dr. Rudolf Diesel, of Munich, Germany, and introduced into the United States under patents and licenses obtained from Dr. Diesel by Mr. Adolphus Busch, of St. Louis, and built by the Busch-Sulzer Bros.-Diesel Engine Company, of St. Louis, in the shops of the Power & Mining Machinery Company, Cudahy, Wis., and installed a little more than a year ago in the plant of the Hugo (Okla.) Ice & Light Company.

Since most that has been written in this country about performances of the Diesel engine has been based upon the showing of European tests, the results of this American test of an American built Diesel heavy oil engine, designed on lines to meet the best ideals of American practice, are interesting.

The test was made on the 225-hp Diesel engine mentioned, by Prof. A. C. Scott, of the Scott Engineering Company, Dallas, Tex., while the engine was about its daily work in the operation of the power plant of the Hugo (Okla.) Ice & Light Company.

The engine had been in service something over six months when the test was made and there was no tuning up, or special preparation made therefor. The test extended over a period of three days and records were taken at all stages from no load to 10 per cent overload.

These records show a fuel consumption of 10.8 gal. of oil at ¼ load; 6.8 at ½ load; easing down to 6.2 gal. at full load for 100 net brake hp-hours, or, 0.441 lb. of oil per net brake hp-hour, and a shade less at the 10 per cent overload.

The thermodynamic records taken, based on net useful output, show 17.4 per cent at ¼ load; 27.8 per cent at ½ load; 29.5 per cent at ¾ load; 30.3 per cent at full load, and 30.2 per cent at 10 per cent overload.

There are no European records known to us which show better, and this speaks well for the American practice.

There have been installed so far over 60,000 hp Busch-Sulzer-Diesel oil engines in various lines of industry in twenty-five states and the Busch-Sulzer Bros.-Diesel Engine Company has now determined on building a large manufacturing plant at St. Louis, to add to the output of the American & British Manufacturing Company, at Providence, R. I., and the Power & Mining Machinery Company, at Cudahy, Wis., who have been building these prime movers for the Diesel Engine Company for some years.

This plant is now nearly completed, and will begin operations about June 1 next. It will, when finished and equipped, represent an outlay of over $1,000,000, and will be one of the most up-to-date, and the largest plant in the United States devoted exclusively to the building of one type of engine.

Gould, Free and Ash is the name of a new firm of chemical engineers with offices in the Monadnock Building, San Francisco, Cal. Mr. Ralph A. Gould was formerly chief of the San Francisco Food and Drug Inspection Laboratory of the United States Bureau of Chemistry. Mr. Edward E. Free was formerly with the United States Bureau of Soils in Field Potash Investigations. Mr. Charles S. Ash was formerly chemist and assistant superintendent of the California Wine Association.

Evaporators for Export

One practice of the Swenson Evaporator Company, of Chicago, which deserves notice has to do with the shipment of goods to countries far distant from the place of manufacture. Two Swenson evaporators which were sent recently to South America on the order of Liebig's Extract of Meat Company, of London, England, were designed for the greatest possible simplicity in construction and operation, thus resulting in a minimum of attention and repairs.

At the same time, with each equipment were sent ten extra tubes, an extra vacuum gage, a full extra set of tube gaskets, extra peep-hole glasses, and additional special gaskets of all kinds. It is a practice of the Swenson Company to furnish these spare parts for foreign shipments because of the distance from the factory and the long time which would ordinarily be consumed in obtaining spares when for any reason they might be needed.

With the pumps were also sent a number of spares, for the same reason.

German Competition on the Smoke Problem

Since the results of the prize competition of Aug. 7, 1908, of the Department of Finance of Saxony as to the prevention of damage by smoke to agriculture and forestry has not been satisfactory, the department has decided to grant rewards for inventions which make it possible to render harmless to plants the gases coming off from furnaces and chemical works without in any way limiting the operation of the works.

Methods and apparatus which are only intended to produce a soot-free combustion will not be considered.

All applications will be investigated and an opinion given on them by the commission appointed by the Department of Finance for the investigation of the smoke evil.

Applications, which must be in German and which must contain any necessary drawings and analyses, should be addressed to the Königlich Sächsische Finanz-Ministerium, II. Abteilung, Dresden, Germany.

Rewards may also be granted for literary work intended to promote the solution of this problem.

The Smoke Abatement Problem

The Committee for the Investigation of Atmosphere Pollution, appointed at the International Smoke Abatement Conference and Exhibition held in London last March, has held three meetings in London and has just published what may be regarded as an "Interim Report."

This report states that after careful consideration of all the various methods that have been suggested or tried for measurement of the impurities of the atmosphere, that employed for the *Lancet* investigation of the soot and dust fall of London in 1911, has been selected as the simplest, and the one most likely to yield satisfactory results, under the conditions which will govern the observations that are to be made. The method is based upon the use of an apparatus resembling an enlarged rain-gauge, with a catchment area of 4 sq. ft. This gauge receives all the dust and soot that falls by its own weight, or is carried down by the rainfall during the period of its exposure, and on examination of the water which collects in the bottle attached to the apparatus, the amount of total suspended matter, tarry oils, soot, etc., can be determined.[1] A circular letter has been sent out by the committee to all the more important city and local authorities in the United Kingdom asking for their co-operation in the application of this method of observation in districts over which they have administrative powers.

This circular has met with a most gratifying response. The authorities of a large number of important cities have already signified their intention of commencing observations on the lines suggested by the committee, and many other authorities are only waiting for further details, before promising their

[1]We intend to describe the apparatus more in detail in a future issue.

support to the movement, and also co-operation in the work.

Birmingham, Bradford, Leicester and Newcastle are the most important of the cities that have definitely promised their support; but there is no doubt that Glasgow, Liverpool, Manchester and London will join in these observations.

The new movement that has been initiated by the committee for studying and recording the character of the soot fall in various industrial centers of the United Kingdom, is therefore meeting with considerable support, and there is little doubt that the observations and records will prove of considerable value to all interested in the progress of smoke abatement.

Dr. W. N. Shaw, F. R. S., of the Meteorological office, is chairman of the committee; and its secretary is Dr. J. S. Owens, 47 Victoria Street, S. W., from whom any further particulars regarding the work of the committee can be obtained.

Annealing Furnace and Electric Steel Furnace

The Midvale Steel Company of Philadelphia, Pa., has ordered a battery of six large Stobie annealing furnaces for the heat treatment of tool steel bars. The installation will be the largest in the world for this class of work, having a total capacity of 7500 tons of steel bars per year.

It is about six years since Mr. Victor Stobie, of Sheffield, erected in that town the first of the then, newly invented furnace for annealing tool steel bars, which had been designed by him. Since that date, in addition to the many furnaces working in Sheffield, a considerable number of furnaces has been erected in steelworks in all parts of Germany and Sweden, including the biggest Continental steel concerns such as Krupps, of Essen, Baildonhütte, of Kattowitz, Lindenburg, of Remscheid, Bismarckhütte, of Schwientochlowitz, Oberschlesische Eisen, A. G., of Gleiwitz, Hofors, A. B., of Sweden, etc.

At the plant of the Midvale company the furnaces are producer-gas fired, the gas being manufactured in plant attached to the main structure. Such gas costs about 1.5 cents per 1000 cubic feet and offers the advantage of being under as good control as a gas kitchen range. The furnaces are built underground, and can be regulated in fifty to one hundred sections according to the length of each furnace.

A few notes should be added on some recent work by Mr. Victor Stobie on electric steel. As has already been noticed in this journal, a big all-electric steel works is being erected on Tyneside by Mr. Stobie which will be all British from foundation to roof. The electric furnace has been designed by Mr. Stobie to "embrace all the features which are looked upon as essential by the expert Sheffield steel maker." No details of construction are yet available. But it is stated that a 5-ton Stobie electric steel furnace is shortly to be erected in a German steel works.

Pulverizing Coal

Elsewhere in this issue will be found a very interesting article by Mr. H. G. Barnhurst, of the Fuller Engineering Company, of Allentown, Pa., on the proper methods of using powdered coal as fuel.

This subject is attracting very considerable attention at present as a result of the rise of the price of crude oil, and in this connection the matter of the cost of pulverizing coal becomes important. Any figures from practice on the cost of pulverizing coal should, therefore, be welcome.

We are obliged to the Bradley Pulverizer Company, of Boston, Mass., for the following figures obtained with Griffin mills in cement plants:

The 30-in. Griffin mill has been used for the past 24 years as a coal pulverizer. The "Giant Mill" has been on the market but three years, but in the short period has demonstrated that it is a very economical mill. It is really an enlarged 30-in. mill designed with the view of giving a larger output and has many improvements which past experience has advised. The Giant mill is now used in many important cement plants for grinding cement rock, slag, cement clinker, and coal.

At the plant of the Penn Allen Cement Company, Penn Allen, Nazareth, Pa., where 30-inch Griffin mills are used, the horsepower consumed is from 18 to 20 hp, the fineness 95 per cent through 100-mesh screen, the output approximately 1½ tons per hour, the cost of up-keep about ¼ cent per ton.

At the plant of the Cape Girardeau Portland Cement Company, Cape Girardeau, Mo., where 30-in. Griffin mills are used, the horsepower consumed is 25 to 30 hp, the fineness 93.2 per cent through 100-mesh, the output 2.67 tons per hour.

At the plant of the Iola Portland Cement Company, Iola, Kan., where 30-in. Griffin mills are used, the horsepower consumed is 25 hp, the fineness 95 per cent through 100-mesh, the output better than 1½ tons per hour.

At the plant of the Knickerbocker Portland Cement Company, Hudson, N. Y., where Giant Griffin mills are used, the horsepower consumed is from 40 to 50, the fineness 94 per cent through 100-mesh, the output over 3 tons per hour, the upkeep "practically nothing."

One of the largest users of Giant Griffin mills for pulverizing coal states that they are procuring an output of from 3 to 5 tons per hour to a fineness of from 90 to 95 per cent through a 100-mesh screen depending on dryness and texture of coal, since, of course, the output of any mill is regulated by dryness and character of material.

Calendars, Diaries, Directories

Calendar.—We have received an attractive calendar from the Lead Lined Iron Pipe Company, which shows several pictures of their lead-lined iron pipe and lead-lined iron valves. This calendar also gives some interesting information about a few of the many installations of their pipe and valves. It will be mailed to interested parties.

American Cement Directory.—We have received from the Bradley Pulverizer Company, Boston, Mass., their very useful American Cement Directory for 1913, compiled by the Technical Press of Boston, Mass. Besides illustrated notes on the Griffin mills, made by the Bradley Pulverizer Company, it contains an alphabetical list of Portland cement plants in operation in this country with the names of officers.

The Badger Chemical Diary is a new annual published by the E. B. Badger & Sons Company, of Boston, Mass., which should prove exceedingly welcome to chemical engineers. Besides illustrated notes on the various chemical apparatus made by this company (evaporation, vacuum pans, stills, dryers, autoclaves and digestors, lead-and-silver-lined apparatus, etc.), it contains useful engineering data on specific heats, expansion coefficient of liquids, properties of various chemicals, strength of materials, properties of saturated steam, heating data on various fuels, conversion tables, etc. It also contains a set of maps and a diary. The intention is to enlarge the engineering data in future issues. But as it is, this first edition marks an excellent beginning.

Personal

Mr. **M. W. Atwater**, of Butte, Mont., has recently resigned his position as general manager for the Butte & Superior Copper Company, to accept the management of the Davis-Daly Copper Company. He succeeds Mr. **William B. Fisher**, who has resigned.

Mr. **James L. Bruce**, general manager for the Continental Zinc Company, Joplin, Mo., is assisting Mr. J. R. Finlay in the examination of the St. Joseph and Doe Run lead mines in southeastern Missouri.

Mr. **C. F. Buck** has been appointed chief engineer and superintendent of construction for the Dominion Nickel Copper Company, and will have charge of the design and construction of a new smelter at Sudbury, Ont. Mr. Buck built the Hayden smelter of the American Smelting & Refining Company, and rebuilt the Tacoma plant of the same company.

The Colorado Chapter of the American Mining Congress, at its annual meeting, elected the following officers and direc-

tors for 1913: Governor, S. D. Nicholson, Leadville; first lieutenant governor, J. M. McClave, Denver; second lieutenant governor, A. L. Burris, Cripple Creek; third lieutenant governor, Bulkeley Wells, Telluride. Executive Committee: V. C. Alderson, Golden; R. L. Martin, Central City; Thomas Tonge, Denver; secretary and treasurer, E. L. Wolcott, 725 Majestic Building, Denver. Directors: D. W. Brunton, Denver; E. W. Bosco, Creede; E. A. Colburn, Denver; R. D. George, State Geologist, Boulder; A. W. Harrison, Silverton; R. M. Henderson, Breckenridge; Irving Howbert, Colorado Springs; W. E. Renshaw, Idaho Springs; John T. Roberts, Jr., Ouray; G. S. Wood, Cripple Creek.

Mr. **Karl Eilers** is making a tour of inspection of the smelters of the American Smelting & Refining Company, and expects to return to New York about the last of March.

Mr. **Robert M. Keeney** has been appointed electrometallurgist of the Bureau of Mines, and will be stationed at Pittsburgh. He will be engaged for the present on experimental work in the treatment of non-ferrous ores in the electric furnace.

Mr. **Frederick Laist** has been appointed general superintendent of the Anaconda smelter of the Anaconda Copper Mining Company, succeeding Mr. William Wraith.

Mr. **Richard K. Meade** has opened an office and laboratory at 202 North Calvert Street, Baltimore, Md. Special attention will be given to lime and cement.

Mr. **Utley Wedge** recently made a trip to the cities on the the Pacific coast.

Mr. William Wraith, who has been general superintendent of the Washoe plant of the Anaconda Copper Mining Company for many years, has been made general manager of the International Smelting & Refining Company, and will reside in Salt Lake City. He assumed his position on the first of this year.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

669,752, March 12, 1901, Paul W. Knauf, of Philadelphia, Pa., assignor to The Electrical Lead Reduction Company, of same place.

Relates to apparatus for electrical reduction of lead sulphide, etc., and consists of a series of preferably cone-shaped pans of antimonial lead, superimposed, so that a gas-tight and insulated joint is made between the several pans. Within each pan is a removable insulating lining, covering the sides, and restricting electrolytic action to the bottom of the pan. The pans are supplied with a suitable quantity of lead sulphide and 10 per cent sulphuric acid, the top pan connected to the positive pole, and the bottom to the negative pole. When a series of such pans are thus connected, the bottom of an upper pan constitutes the anode, and the lead sulphide the cathode; hydrogen liberated by electrolysis reduces the lead sulphide to metallic lead, and forms hydrogen-sulphide which passes out through a vent to a stack.

669,926, March 12, 1901, Carl Hoepfner, of Frankfort-on-the-Main, Germany.

Relates to a method of depositing zinc from a solution in a three-compartment cell. The cell consists of a plurality of U-shaped sections, separated by diaphragms, all bolted together. Every anode and cathode compartment is connected in series by passages, so that the anode electrolyte circulates through anode compartments, and the cathode electrolyte through cathode compartments. The intermediate compartments are also connected by passages, either in series or in parallel. The level of the solutions is so regulated that the catholyte is at a higher level than that of the intermediate cell, and still higher than that of the anolyte. There is therefore a constant diffusion of solution from the cathode compartment toward the anode compartment, which prevents contamination of the catholyte by the anolyte. The anode is a metal soluble in the anolyte, and electronegative to zinc, as lead or copper. On dissolving, it forms nitrates, acetates, etc., according to the electrolyte used, and after circulating through the several anode-compartments, is precipitated in a suitable vessel outside of the cell, by a zinc salt as zinc sulphate, zinc carbonate, etc. The solution freed of lead, etc., is now used as the catholyte. The intermediate compartment is supplied with a solution preferably identical with the catholyte, and contains a steam-pipe which is electrically connected to the cathode through a resistance. As anodes, impure lead or copper ingots may be used, and the metal subsequently precipitated is highly pure and may be suitably reduced or otherwise treated. Any lead or copper that diffuses into the intermediate compartment will be precipitated by electrolysis; or the intermediate electrolyte and catholyte may be further purified by zinc dust. All of the electrodes are constantly vibrated to assist stirring and depolarization.

677,089, June 25, 1901, John C. Kessler, of Milwaukee, Wis., assignor of one-half to Christian Wahl, of same place.

Relates to the extraction of native copper from ores or tailings by a leaching solution containing a cupric ammonium compound, such as cupric oxide, cupric sulphate, etc., dissolved in ammonia. The tailings or pulverized ore is placed in a leaching tank and covered with the cupric ammonia solution; this is then covered with about 4 in. of water which prevents loss of ammonia by evaporation. After 24 or 48 hours, the lower ammonia solution is run off, and the ore covered with water to wash it. The cupric ammonia solution dissolves copper forming cuprous-ammonium compounds, as follows:

$$CuO.2NH_3 + Cu = Cu_2O.2NH_3.$$

This solution thus obtained is electrolyzed, precipitating half of the copper, the remainder being oxidized to the cupric state and reused.

678,526, July 16, 1901, Charles P. Stewart, of Oakland, Cal.

Relates to an apparatus in which gold, etc., is electrolytically deposited from a flowing cyanid solution upon a quiescent body of mercury. The apparatus consists of a long shallow trough or tank through which a cyanid solution flows. The bottom of the trough is covered with a layer of mercury electrically connected as a cathode; a number of separate anodes are adjustably suspended in the solution, the adjustment providing for any irregularities in wear. When the mercury is saturated, it is removed and the precious metals separated in any suitable manner.

682,155, Sept. 3, 1901, Charles P. Tatro and George Delius, of Seattle, Wash.

Relates to an apparatus for extracting precious metals, and refers to their prior patents 640,718 and 653,325. This apparatus consists of a suitable acid-proof tank having a copper pan covering the bottom, and a mercury cathode in the copper pan. Above the mercury are rotating paddles which keep the pulp and solution well stirred. Above the paddles are a number of depending carbon anodes, supported by a lid; the lid, anodes and paddles are capable of being raised vertically from the tank to facilitate charging, etc. A supplementary anode projects through the side of the tank and serves to keep current on, thereby electrolytically protecting the mercury and copper from corrosion when the main current is cut off. When the operation is completed, the tank is emptied through a port in the bottom.

689,018, Dec. 17, 1901, William Orr, of Salt Lake City, Utah, assignor to The Gold and Silver Extraction Company of America, Ltd., of Denver, Col., a joint stock company of Great Britain.

Relates to a method of regenerating cyanid solutions, particularly those containing copper and zinc cyanids, and an apparatus to electrolytically precipitate the copper. The apparatus consists of a tank having a plurality of pairs of electrodes, alternate ores of zinc connected as anodes, and the intermediate

ones of copper, connected as cathodes. The solution flows between the electrodes, the copper being deposited and an equivalent of zinc going into solution. To the solution now containing zinc cyanid is added an alkaline or alkaline earth hydroxid, forming, for example, sodium zincate. To this solution is added just enough sodium sulphide to form zinc sulphide which separates, and may, if desired, be filtered off and recovered. The solution now consists of the original cyanid plus some alkaline or alkaline earth hydroxid, and is used in the usual manner for treating ores.

689,070, Dec. 17, 1901, Alexander Stanley Elmore, of London, England.

Relates to the separation of mineral substances by the selective action of oil, and states that in the presence of an acid the separation is more effective. The apparatus used consists of a mixer, a settling tank, and a centrifugal separator. The ore is suitably powdered and is supplied to the mixer with five to ten times its weight of water, forming a thin freely flowing pulp. To the mixer is also added a thin stream of oil, and an acid soluble in oil but insoluble in water, such as oleic acid which together with the oil coats the metallic particles, such as metal dust, tellurids, sulphides, graphite and elemental sulphur, with a film of oil, thereby causing them to float. Instead of oleic acid, or a similar acid, an acid soluble in water but insoluble in oil, such as sulphuric acid, may be used. The quantity of acid added, in either case, will vary with the materials being separated; in some cases it may be as little as 1/500 part of the oil or water used.

689,674, Dec. 24, 1901, Albert Isaiah Irwin, of Cripple Creek, Col., assignor of one-third to Caleb F. Bryant, of Cripple Creek Mining District, Teller County, Col.

Relates to apparatus for electrolytically dissolving and precipitating the metallic content of an ore. The electrolytic apparatus consists of a traveling endless belt, of metal, on which are cross-pieces of insulating material. The movement of the belt conveys the ore through a suitable solution, and discharges it from the machine. The belt is connected as an anode, and adjacent thereto are cathodes. With gold ores, a cyanid solution may be used as electrolyte, the dissolved gold and silver being electrolytically deposited. If the ore gives an acid reaction to water, sufficient alkali is added to maintain the alkalinity of the electrolyte.

689,959, Dec. 31, 1901, Edward Leslie Graham, of Upper Warlingham, England.

Relates to the disintegration and comminution of ores, etc., by the action of gases generated by passing an electric current through a suitable acid solution, such as sulphuric, hydrofluoric, or mixed sulphuric and hydrofluoric. Other acids and salts may also be added, such as ammonium chlorid or carbonate, barium chlorid, fluorspar, etc. With free-milling gold ores, 2½ per cent of sulphuric acid and 1 per cent of hydrofluoric acid are said to give good results; other ores, such as those of copper, silver, and antimony, require different percentages. The current strength will vary with the ore and size of the vat, and should have a voltage of about fifty volts per vat.

Book Reviews

Die Spezialstähle. Ihre Geschichte, Eigenschaften, Behandlung und Herstellung. By **G. Mars,** Dipl.-Ing. Octavo (15½ x 24 cm.), 517 pages, 143 illustrations; price 17 marks, bound 18.40 marks. Stuttgart: Ferdinand Enke.

A monumental treatise by the director of the research department of the Rhenish Metalware & Machinery Works, in Düsseldorf. It is far more than a mere compilation, for the author discusses in a very instructive manner the relations of the steels to each other, the bearing of the periodic system of the elements on the effects produced by different metals, and even reflects upon the somewhat neglected question of the improvement of these steels by proper treatment of the fluid metal.

After chapters on history, metallography, the iron-carbon diagram, heat treatment, forging and cold-working of steels, the author takes up in distinct chapters the carbon, Si, Mn, Cr, Mo, W, V, Ti, Al, and Ni steels, and more briefly, in two chapters, the V, Ta, B, Sn, Co and Cu steels. There are also separate chapters on construction steel, high-speed tool steel, and (76 pages) on the manufacture of the special steels.

The whole is worked over with great accuracy and completeness. Friends of the electric furnace steel will read with satisfaction the verdict of this expert upon the *rôle* of the electric furnace in the manufacture of these special steels: "The electric steel smelting process appears to be undoubtedly the highest stage in the development of steel manufacturing processes."

* * *

Metallurgische und technologische Studien über das Ausglühen von Metallen und Legierungen. Von Dr. Ing. **Max Weidig.** Large quarto (22½ x 30 cm.), 121 pages, 48 illustrations; price 6 marks. Berlin: Leonhard Simion Nf.

The author is privat-docent at the Bergakademie, Freiberg. In one-half of the book he reviews the results of others on the influence of annealing upon the structure and mechanical properties of metals and alloys; in the second half he relates his own experiments upon nine materials: copper, nickel, copper-nickel, brass, aluminium bronze, tin bronze, tin-zinc bronze, german silver, and aluminium. Each of these materials were tested hard and annealed, and the effects on color, luster, structure, hardness, tensile strength, ductility, and electrical conductivity carefully determined.

An interesting observation is the relative quantities of heat energy absorbed in annealing slowly at a low temperature, and quickly at a higher temperature. Using an electric furnace for heating, it is usually more economical of power to anneal quickly at the higher temperature. This is only one of many useful items of information in this monograph.

* * *

Transactions of the American Foundrymen's Association: Vol. xx. Edited by **Richard Moldenke,** Secretary. Octavo (15 x 23 cm.), 684 pages, numerous illustrations. Watchung, N. J.: The Association.

A fine collection of mostly short papers, dealing with details of foundry practise up to the theory of alloys. An 18-page paper, finely illustrated, on the use of the induction furnace for making steel castings, by Mr. C. H. Vom Baur, is a notable contribution to that subject; a paper on pyrometry, by Mr. S. H. Stupakoff, also deserves mention. Many of the others are well worth reading, although the English and the manner of presentation are frequently faulty. The printing and binding are excellent.

* * *

Die Praxis des Eisenhüttenchemikers. By Dr. **Carl Krug.** 15 x 23½ cm., 226 pages, 31 illustrations; price 6 marks. Retail price New York, $2.00. Berlin: Julius Springer.

Chemische Untersuchungsmethoden für Eisenhütten und deren Nebenbetriebe. By Ing. **Albert Vita** and Dr. **Carl Massenez.** 13 x 21 cm., 175 pages. Price 4 marks. Retail price New York, $1.35. Berlin: Julius Springer.

Dr. Krug is "Dozent" in the "Bergakademie" in Berlin. His book is the product of 15 years' teaching experience, and is very carefully and painstakingly written. Engineer Vita is chief chemist for the Friedenshütte in upper Silesia, and Dr. Massenez teaches analysis in the Technical High School of Breslau. Their book is the product of practical experience, omitting all methods which have been superseded in recent years.

Both works are commended to the iron and steel works' chemist. Science is the most truly international factor in modern life, and the scientific man has the widest opportunity to become cosmopolitan in his profession if he will only avail himself of such assistance and information as are at his disposal. Thus, a mere bagatelle of ten marks will put him abreast of the best European practice in iron and steel works' laboratories.